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RESOURCE DEVELOPMENT EVALUATION

of the

GARFIELD COAL DEPOSIT

Garfield and Kane Counties, Utah

Submitted to
U.S. DEPARTMENT OF THE INTERIOR

by
UTAH POWER AND LIGHT COMPANY
Salt Lake City, Utah

Prepared by
SKELLY AND LOY, Engineers and Consultants
Denver, Colorado - Harrisburg, Pennsylvania
COAL SYSTEMS, INC. - Salt Lake City, Utah
NUS CORPORATION - Denver, Colorado
JOHN PEPERAKIS, P.E. - Salt Lake City, Utah
and
UTAH POWER AND LIGHT COMPANY
Mining and Exploration Department

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1.0 SUMMARY

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1. Summary

Introduction 1.1

Utah Power & Light Company (UP&L) acquired rights to Prospecting Permits encompassing 18,325.2 acres of federal coal land and State of Utah coal leases comprised of 5,119.96 acres of contiguous and nearby land in Garfield and Kane Counties of Utah in 1971 (Figure 1-1). Since that time, UP&L has diligently pursued a course that would result in the development of the coal reserves in this coal acreage, hereinafter referred to as the "Garfield Deposit", in strict accordance with federal and state requirements. To date, these actions have culminated in the negotiation of a lease exchange agreement that permits UP&L and the Department of Interior (DOI) to exchange a portion of UP&L's rights to federal coal lands in the Garfield Deposit for federal coal lands of equal value in central Utah.

Consistent with the agreement between UP&L and DOI that was consummated on July 30, 1979, UP&L has prepared a technical and economic evaluation concerning the Garfield Deposit that permits the definition of the value of the deposit for exchange purposes. The following chapters of this report summarize, in consecutive order, the geology, mining plans and surface facilities, railroad transportation, and economics relative to the development of the Garfield Deposit.

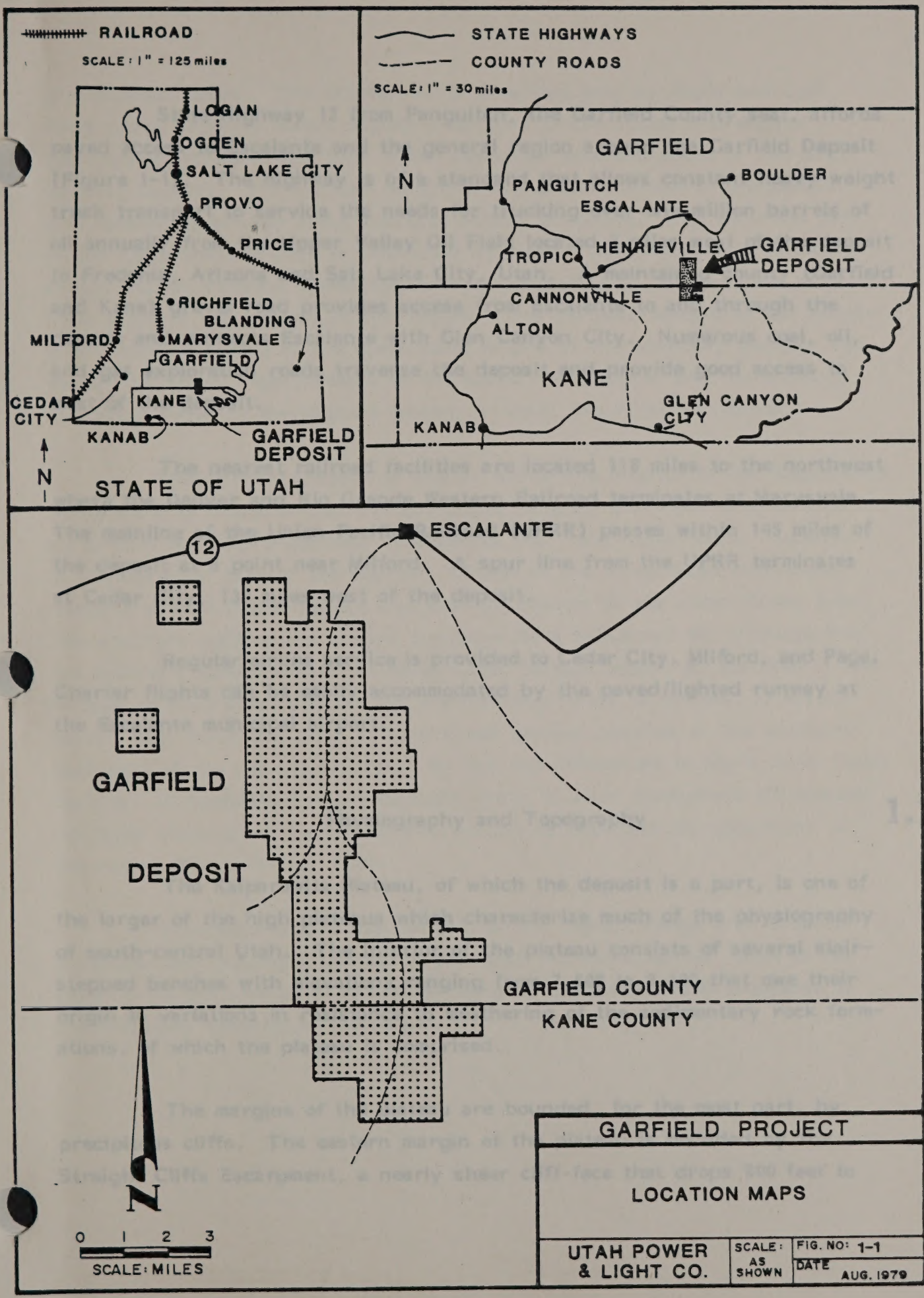
Location and Access 1.11

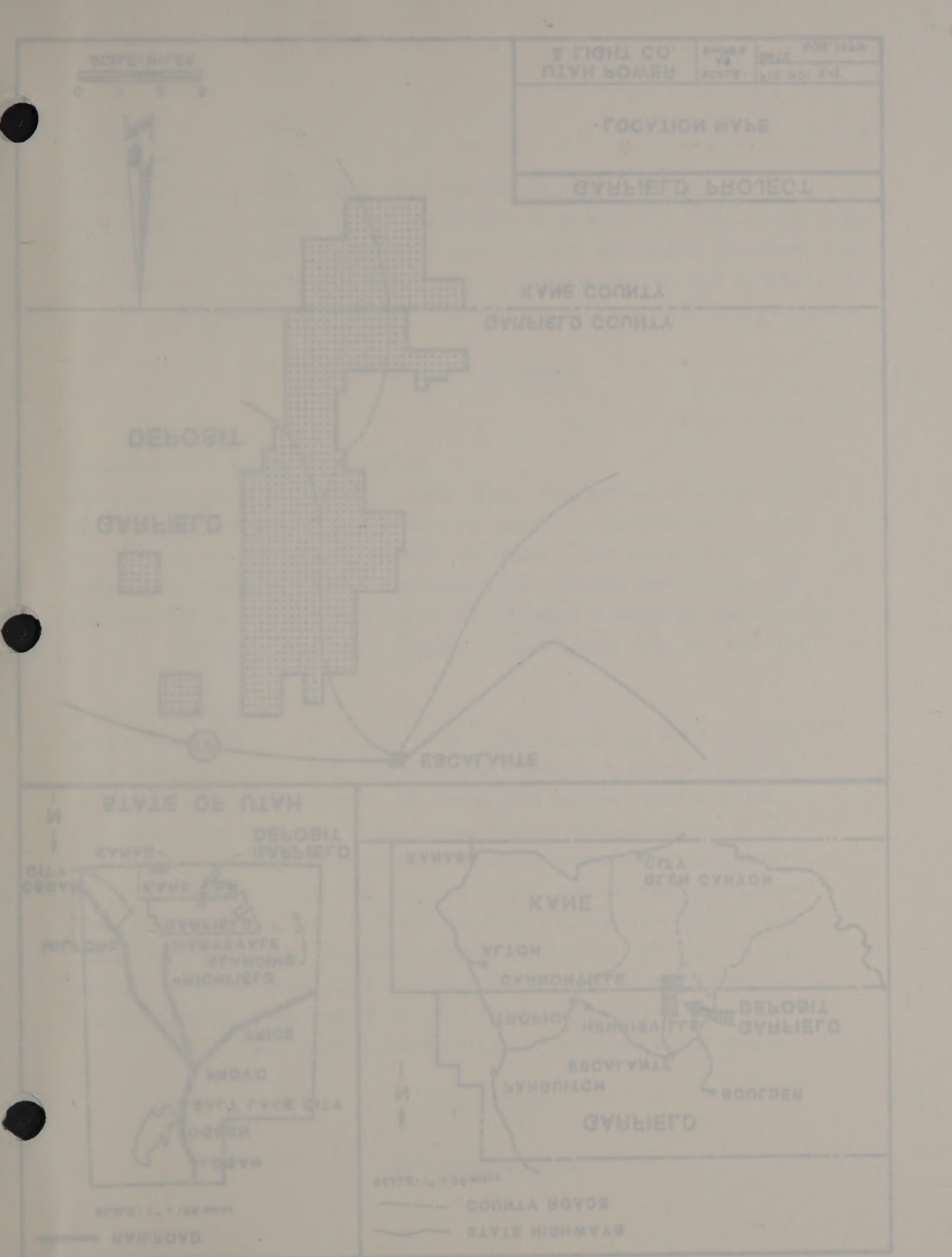
The Garfield Deposit is located in the northern part of the Kaiparowits Plateau in Garfield and Kane Counties of south-central Utah (Figure 1-1). The nearest community, Escalante, a town of about 600 people, is located nine miles north of the deposit. The communities of Boulder (93), Henrieville (145), Cannonville (113), and Tropic (329) are located within a 30 mile radius of the deposit.

1. Summary

Introduction

1.1


RAILROAD
SCALE: 1" = 125 miles
LOGAN
OGDEN
SALT LAKE CITY
PROVO
PRICE
RICHFIELD
BLANDING
MILFORD
MARYSVALE
GARFIELD
CEDAR CITY
KANE
KANAB
GARFIELD DEPOSIT
N
STATE OF UTAH
STATE HIGHWAYS
COUNTY ROADS
SCALE: 1" = 30 miles
GARFIELD
BOULDER
PANGUITCH
ESCALANTE
TROPIC
HENRIEVILLE
GARFIELD DEPOSIT
CANNONVILLE
ALTON
KANE
GLEN CANYON CITY
KANAB
ESCALANTE
12
GARFIELD
DEPOSIT
GARFIELD COUNTY
KANE COUNTY
N
0 1 2 3
SCALE: MILES
GARFIELD PROJECT
LOCATION MAPS
UTAH POWER & LIGHT CO.
SCALE: AS SHOWN
FIG. NO: 1-1
DATE AUG. 1979



State Highway 12 from Panguitch, the Garfield County seat, affords paved access to Escalante and the general region around the Garfield Deposit (Figure 1-1). The highway is of a standard that allows constant heavy-weight truck transport to service the needs for trucking over one million barrels of oil annually from the Upper Valley Oil Field located 8 miles west of the deposit to Fredonia, Arizona and Salt Lake City, Utah. A maintained county (Garfield and Kane) gravel road provides access from Escalante to and through the deposit and connects Escalante with Glen Canyon City. Numerous coal, oil, and gas exploration roads traverse the deposit and provide good access to most of the deposit.

The nearest railroad facilities are located 118 miles to the northwest where the Denver and Rio Grande Western Railroad terminates at Marysvale. The mainline of the Union Pacific Railroad (UPRR) passes within 145 miles of the deposit at a point near Milford. A spur line from the UPRR terminates at Cedar City, 130 miles west of the deposit.

Regular airline service is provided to Cedar City, Milford, and Page. Charter flights can be easily accommodated by the paved/lighted runway at the Escalante municipal airport.

Physiography and Topography 1.12

The Kaiparowits Plateau, of which the deposit is a part, is one of the larger of the high plateaus which characterize much of the physiography of south-central Utah. The interior of the plateau consists of several stair-stepped benches with elevations ranging from 7,600 to 9,100 that owe their origin to variations in resistance to weathering of the sedimentary rock formations, of which the plateau is comprised.

The margins of the plateau are bounded, for the most part, by precipitous cliffs. The eastern margin of the plateau is bounded by the Straight Cliffs Escarpment, a nearly sheer cliff-face that drops 800 feet to

the flatlands below. The escarpment is breached only by Right and Left Hand Collet Canyons whose tributary canyons deeply incise and expose coal-bearing strata in the area to the immediate east and south of the UP&L Garfield Deposit. The western margin of the plateau is also a precipitous cliff formed by erosion along the East Kaibab Monocline. The southern part of the Kaiparowits Plateau is dominated by several north-northwest trending deeply incised canyons whose alignment is controlled by the dominant structural grain of the geology in the region. The northern margin of the plateau is ill-defined because of the transitional nature of the boundary between the heavily vegetated high plateaus of the Kaiparowits Region and the similarly vegetated slopes of the highest plateau in Utah, the Aquarius Plateau.

The Garfield Deposit is located within the interior of the Kaiparowits Plateau and is characterized by flat-topped benches that have been incised by several northward-draining tributaries that form Alvey Wash. Locally, in the northern part of the deposit, several of the Alvey Wash tributaries coalesce to form broad valleys and wide grasslands in the Little Valley Area. The southern part of the deposit has been more influenced by drainage into the tributary canyons of Right Hand Collet Canyon which has resulted in the incision of several canyons in the coal-bearing strata to the east of the deposit. The result has been to form broad plateau benches in the southernmost part of the deposit separated by the few tributaries to Right Hand Collet Canyon. A compilation of photographs taken at sites throughout the deposit has been included at the end of this section to illustrate the geography of the region (Plates 1-1 to 1-12).

Elevations within the confines of the deposit range from 6,800 to 5,600 feet but generally average 6,000 feet. As noted earlier, most of the relief is abrupt and steep in nature owing to the downcutting processes of the tributary canyons in the region.

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Elevations within the confines of the deposit range from 6,800 to 5,600 feet but generally average 6,000 feet. As noted earlier, most of the relief is abrupt and steep in nature owing to the downcutting processes of the tributary canyons in the region.

Deposit Boundaries 1.13

The 18,325.2 acres of federal Preference Right Lease Application lands consist of 8 Prospecting Permits, numbers U-1362, U-1363, U-1375, U-5233, U-5234, U-5235, U-5236, and U-5237 (Figure 1-1 and Appendix 2-1). Adjacent and contiguous to the federal lands are 5,119.96 acres of coal lands leased from the State of Utah. An additional 1,240 acres of state lands are held by UP&L within five miles of the main block of land holdings.

The federal lands within the deposit are administered by the Bureau of Land Management. The western boundary of the deposits is formed by the Dixie National Forest whose surface is administered by the Department of Agriculture/U.S. Forest Service. The state lands within the deposit are administered by the State of Utah.

Activities to Date 1.14

UP&L acquired the Prospecting Permits and State Leases which comprise the Garfield Deposit in 1971. In accordance with requirements concerning Prospecting Permits, UP&L conducted exploratory drilling programs in 1971 and 1972 to find "commercial quantities and qualities of coal". These drilling programs resulted in the completion of 24 core drill holes on a spacing grid of approximately one hole per section. As the data summarized in the chapter on geology will indicate, substantial quantities of coal of high quality were encountered in all drill holes.

The data from the 24 drill holes indicated that a federal requirement for finding "commercial quantities and qualities of coal" had been met and that UP&L was entitled to a conversion of the Prospecting Permits to Preference Right Coal Leases. Accordingly, UP&L filed for conversion to Preference Right Leases in December of 1972. The application for conversion was not processed and leases were not issued.

New regulations promulgated in 1976 indicated the necessity for submitting all technical data to the DOI concerning the Garfield Deposit in the form of an "Initial Showing". UP&L complied with the requirement and submitted an "Initial Showing" which consisted of a narrative concerning the in-place and recoverable reserves, mining methods, and transportation alternatives as well as numerous drill hole logs, coal quality data, and maps. The "Initial Showing" indicated the presence of at least 400 MM tons of recoverable coal in the deposit. Once these data were submitted, UP&L was requested to submit another report concerning the economics of developing the Garfield Deposit. Again, UP&L complied and submitted a "Final Showing" that indicated that coal reserves could be produced for $13.20 per ton at an annual production rate of 10 MM tons and that railroad transportation rates to a plant site near Nephi would be on the order of $5.00 per ton.

The data presented within the following chapters of this report further refines the data contained within the "Initial" and "Final Showings" and represents the most detailed examination of the geology, mining, surface facilities, transportation, and attendant economics developed for the Garfield Deposit to date. The scenario presented herein is developed within the scope of governmental requirements (diligent development) and is the same general order of magnitude for other development scenarios being entertained by other companies holding coal leases in the Kaiparowits Plateau Coal Field.

Summary of Geology and Coal Reserves

Outcrop mapping activities by U.S.G.S. and UP&L geologists and 24 drill holes completed by UP&L permits the evaluation of the geology and coal reserves of the Garfield Coal Deposit. Data collected to date indicate the following:

- General Geology of the region has been defined by Zeller of the U.S.G.S. and has been summarized in four Geologic Quadrangle Maps.
- Geologic Investigations have been conducted by various workers of the U.S.G.S., academic institutions, state agencies, and exploration companies utilizing conventional mapping and drilling techniques. Stratigraphic principles and the analyses of depositional environments have been utilized in defining the geology of the deposit.
- Geologic Analysis indicated that the coal deposit is essentially flat-lying and is enclosed within the John Henry Member of the Straight Cliffs Formation (Upper Cretaceous) (Figure 1-2). The member is comprised of five major rock types and can be subdivided cartographically and genetically, into seven major regressive sedimentary sequences that owe their origin to pulsations in the growth of deltas. Coal seams are inferred to have been deposited in swamps along the margins of a strandplain/barrier island depositional system during progradational events.
- Three major coal zones developed in response to the three mast widespread regressions which are referred to, in ascending stratigraphic order, as the Christensen, Rees, and Alvey Coal Zones. These zones are comprised of eight, six, and one coal seams, respectively, of which eight are of sufficient lateral continuity and thickness to be considered for mining.

Summary of Geology and Coal Reserves

Outcrop mapping activities by U.S.G.S. and UPSU geologists and 74 drill holes completed by UPSU permits the evaluation of the geology and coal reserves of the Garfield Coal Deposit. Data collected to date indicate the following:

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- Three major coal zones developed in response to the three most widespread regressions which are referred to, in ascending stratigraphic order, as the Christensen, Rees, and Alvey Coal Zones. These zones are comprised of eight, six, and one coal seams, respectively, of which eight are of sufficient lateral continuity and thickness to be considered for mining.

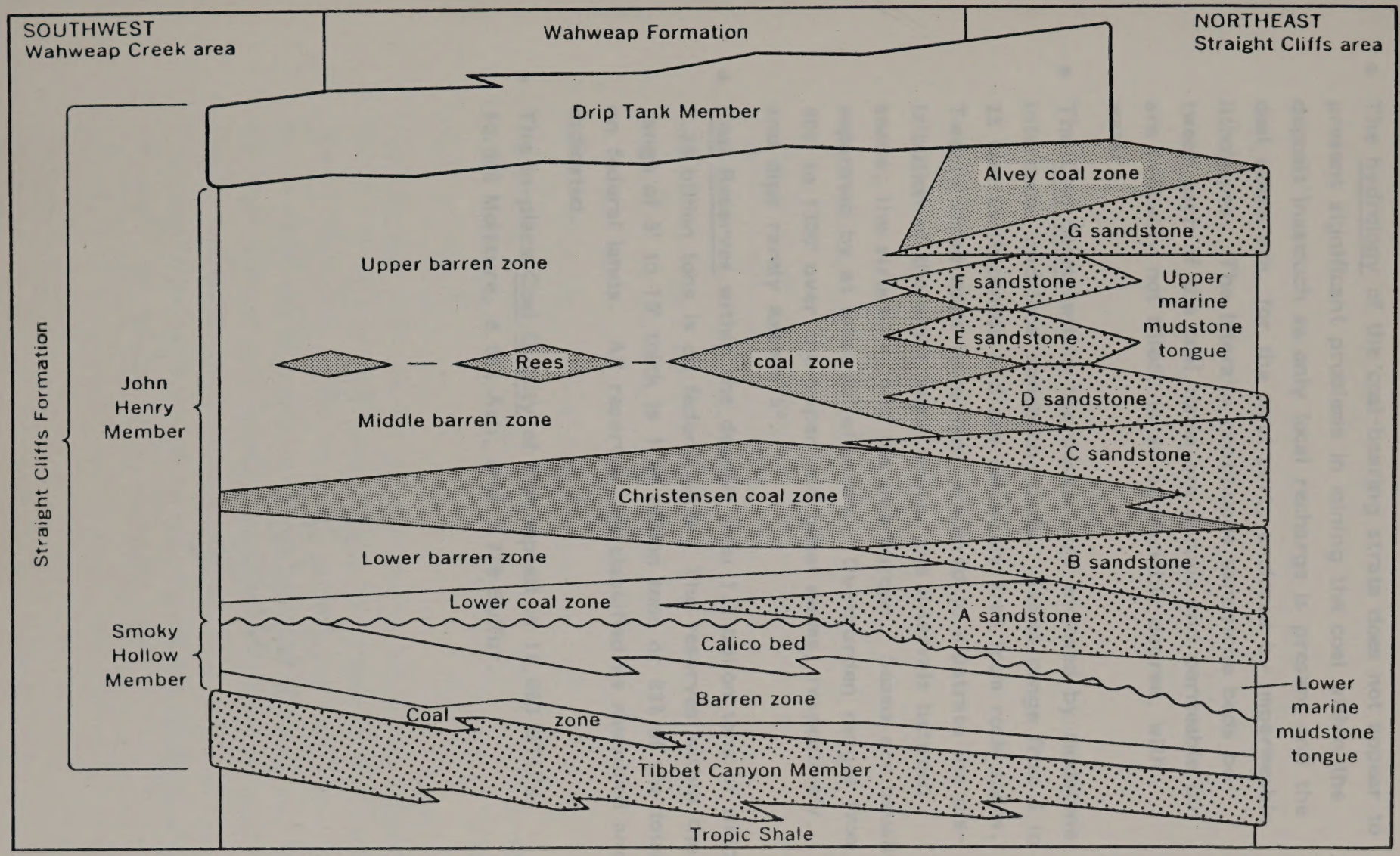
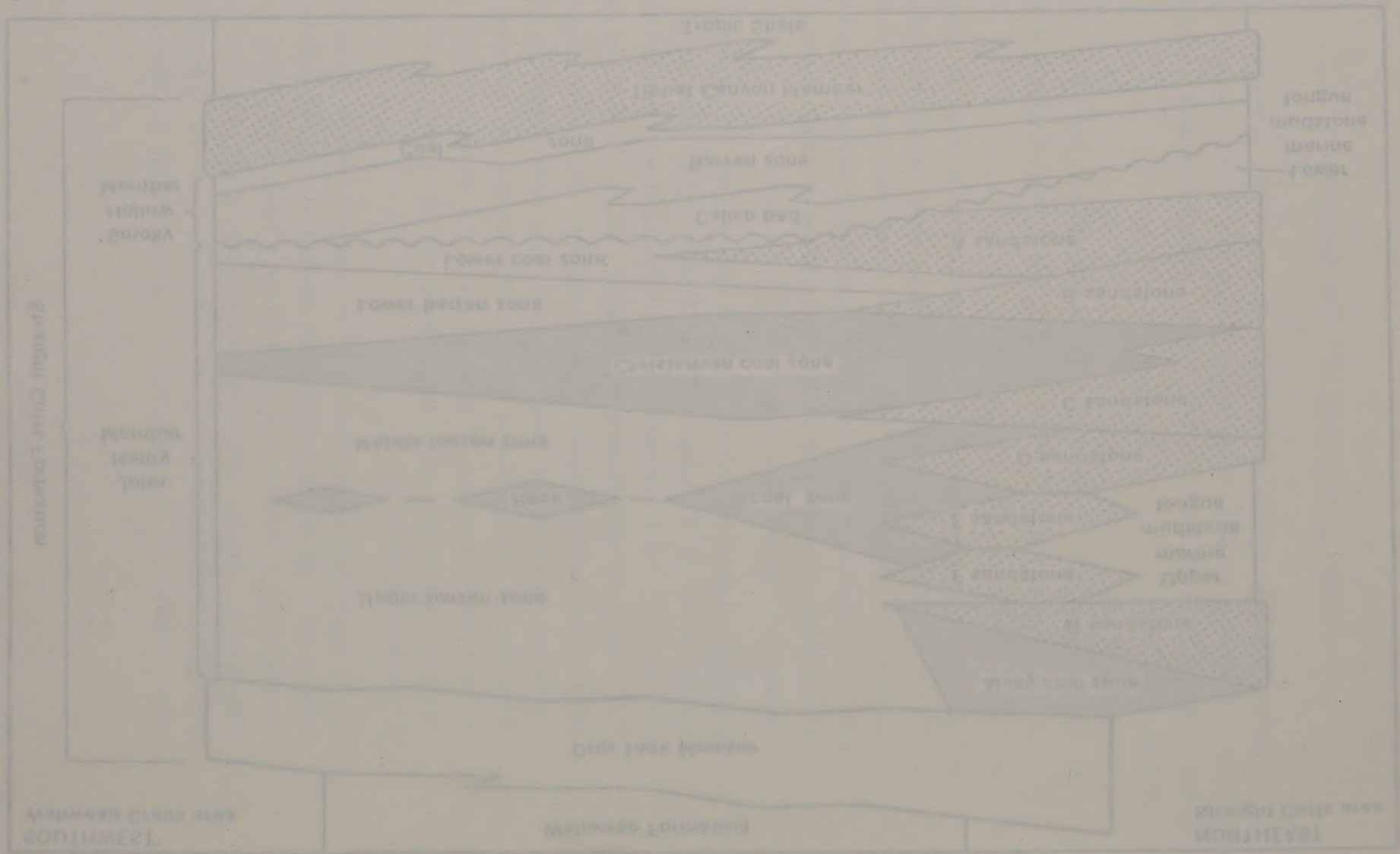


FIGURE 1-2: Generalized cross section through the Kaiparowits Plateau (Peterson, 1969a). The Garfield Deposit is generally located in the right-hand side of the diagram.

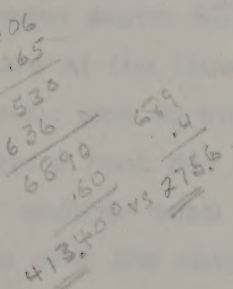


- The hydrology of the coal-bearing strata does not appear to present significant problems in mining the coal within the deposit inasmuch as only local recharge is present and the coal seams are, for the most part, enclosed by impermeable lithologies. The laterally continuous sandstone beds between each of the coal zones are porous and permeable but are probably not filled, to an appreciable degree, with water.

- The Coal Seams within the deposit are enclosed by mudstone, interbeds, and local sandstone beds. Seams range from 5 to 25 feet in thickness and are relatively free from rock splits. Twenty-three maps have been prepared to illustrate the distribution of the eight major seams, the intervals between seams, the structure, and the overburden. Seams are usually separated by at least 40' of strata. Overburden ranges from 900' to 1300' over the upper and basal seams, respectively, and dips rarely exceed 5°.

- Coal Reserves within the deposit total 1.5 billion tons of which 1.28 billion tons is on federal land. The reserves within the range of 5' to 10' thick is 1.06 billion tons or 83% of the total on federal lands. All reserves are classified as measured and indicated.

- The in-place Coal Quality of the deposit is 10,683 BTU's, 10.00% Moisture, 8.06% Ash, and 0.72% Sulfur.



1.3 Summary of Mining Plan

This report substantiates that development and operation of an 18 MMTPY underground mining complex on Utah Power and Light Company's Federal lease application lands in Garfield and Kane counties is feasible. Some of the important features of the mining plan are summarized in Table 1-1.

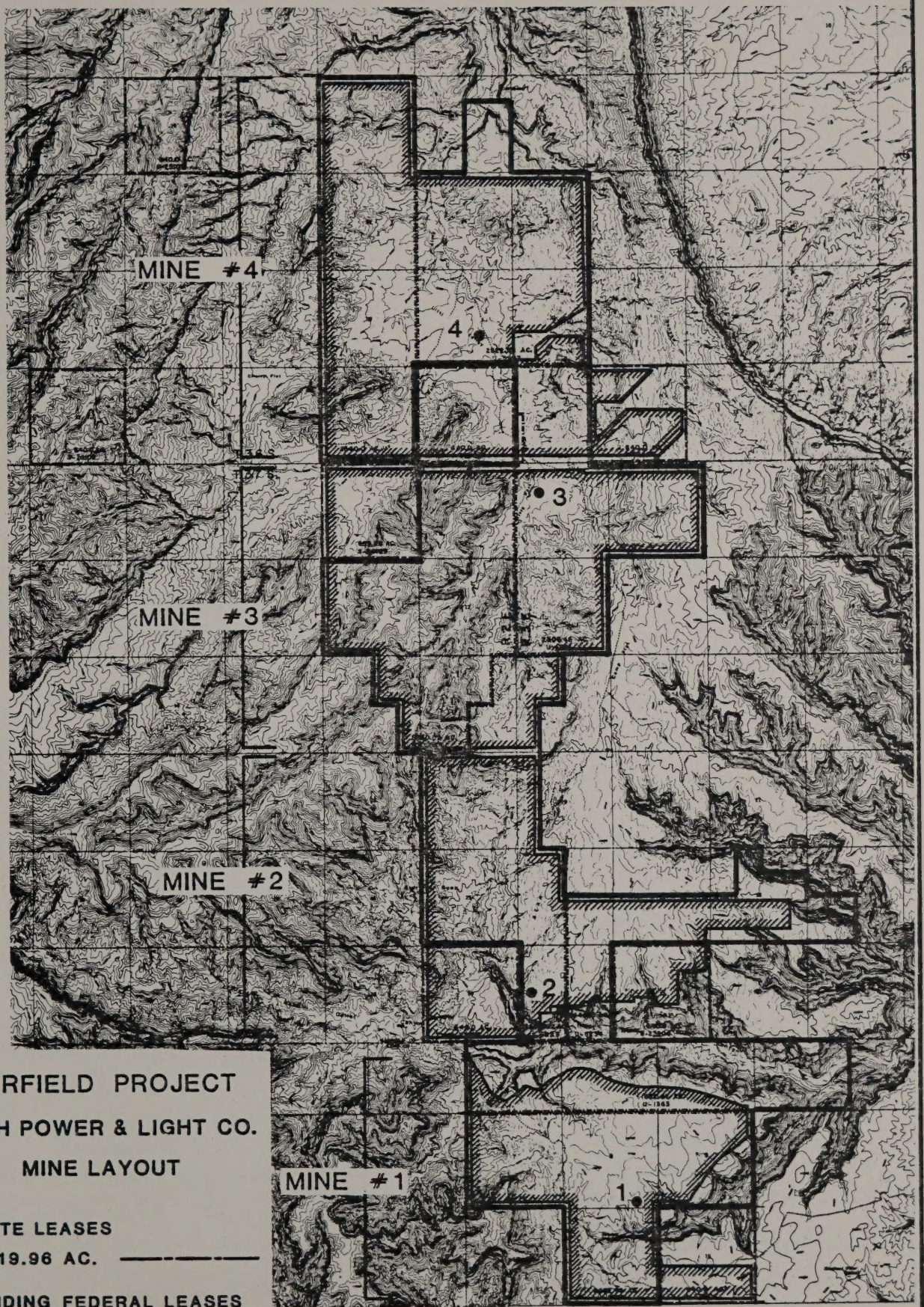
- Three underground mines can be developed to produce 5 MMTPY (21,600 TPD) each, and a fourth mine can be developed to produce 3 MMTPY (12,960 TPD). The boundaries of these mines are shown in Figure 1-3.
- The coal seams can be reached by rock slopes of lengths varying from 2,500 to 4,100 feet which would provide belt coal haulage and rail supply service, as well as intake air.
- The mines can be developed by continuous miners in multiple entry mining producing 330 tons per unit shift, with primary production units by longwall panels producing 1500 tons per unit shift. A total of 18 longwall units and 36 continuous miner units can be installed on a phased basis to reach the required annual production level within ten years of project start-up.
- Multiple-seam underground mining can be done by sequential seam development and extraction progressing downward from the top seam.
- Coal haulage from the underground seams will be accomplished by shuttle cars or face conveyors at the faces, 48-inch conveyors from each section or panel, and 60-inch conveyors in the main entries emptying into a rock bin or raise constructed to lower coal to either another seam level or onto feeder-accelerator belt interface with the main slope belt. The main slope belts will be 60-inch conveyors rising at a 16° angle (28% grade) to the surface.

TABLE 1-1

GARFIELD PROJECT MINE INFORMATION SUMMARY

	Mine #1	Mine #2	Mine #3	Mine #4	Total
Coal Reserves					
Recoverable Reserves (MMT)	107.7	166.8	199.8	137.9	612.2
Annual Production Rate (MMT)	3.0	5.0	5.0	5.0	18.0
Production Units					
Longwall Units	3	5	5	5	18
Continuous Miner Units	6	10	10	10	36
Underground Access					
Slope Lengths (one each)	2560'	4060'	3810'	2920'	13,350'
Shaft Depths	4 @ 705'	5 @ 1100'	5 @ 1050'	6 @ 805'	20/18,400'
Ramp Lengths	none	2 @ 320'	1 @ 520'	1 @ 730'	4/1570'
Surface Belting					
Overland Belt* - Length	6000'	5300'	1800'	10,000'	23,100'
Overland Belt - Width	60"	60"	60"	60"	
Surface Facilities					
Coal Preparation Plant	1	1	1	1	4
Shop/Office/Warehouse	1	1	1	1	4
Central Office/Shop					1
Manpower					
General Staff					62
Mine Staff	71	91	91	91	344
Central & Loadout Staff					25
Mine Wage Earners	420	619	619	619	2277
Central & Loadout Wage Earners					130

*46,300' of 72" belt is shared by Mines 2, 3, and 4 which yields a total of 69,400 feet (13.14 miles)



GARFIELD PROJECT

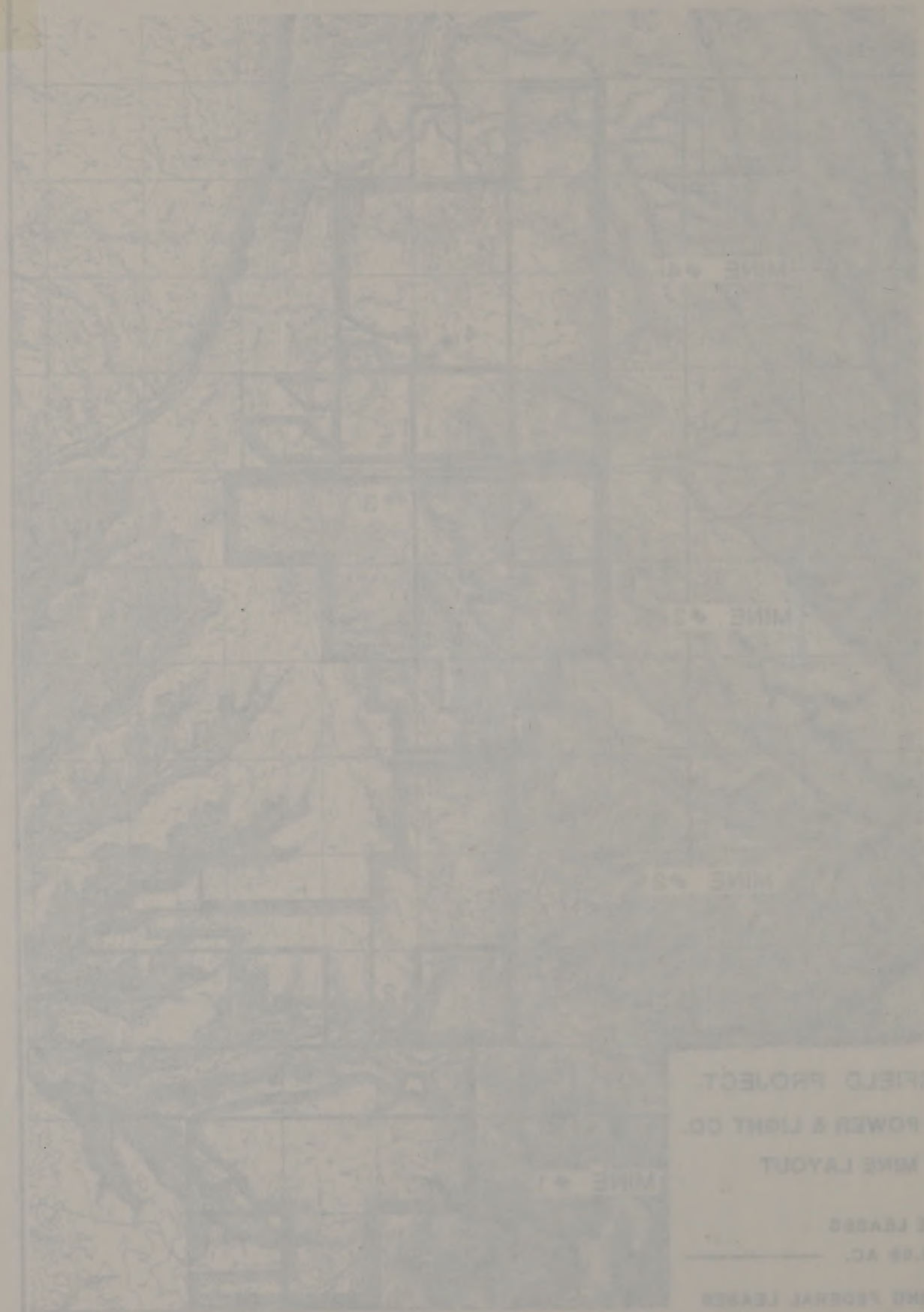
UTAH POWER & LIGHT CO.

MINE LAYOUT

STATE LEASES
5,119.96 AC. ———

PENDING FEDERAL LEASES
18,325.16 AC. ———

FIGURE 1-3



- Each of the four mines will have a coal sizing plant at the slope portal to crush, screen, sample, and deliver coal to the overland belt system. Emergency surge piles will be included at each mine-mouth to allow mine production to continue despite surface belt outages. These mine plants will have 1800 TPH maximum capacity (1200 TPH nominal).

- Coal haulage between the mines to the rail loadout/storage facility will be by 60- and 72-inch overland conveyor belts supported by cable suspension between concrete/steel frames on 40-foot spacing (Figure 1-4). Flexibility and easy re-alignment are required as the overland belt may be affected by subsidence. Transfer points will be contained in permanent structures with protection for the drive motors and dust collection to minimize fugitive dust emissions.

- The coal storage/loadout facility will consist of a barn-and-slot type storage pile of 150,000 ton capacity (two days production), with an emergency storage pile of the double-cone type. Twin reclaim conveyors would run in a tunnel beneath the two storage piles, carrying the coal up to a twin loadout structure. Train loading on either of two parallel loop tracks would be achieved by coal flowing into either of two 200-ton surge bins, then into either of two 100-ton batch bins, then through a chute into the rail cars. The ability to load two trains concurrently is believed to be necessary because of the anticipated 60,000 tons (6 trains) per day shipment level, and the three to four hour loading time required by most tariffs. Perfect sequencing of trains would be required if only one loadout was available, so a duplicate is suggested. Each of the loadout bins would have an automatic sampling system attached to provide the most representative train sample.

- Each of the four mines will have surface facilities at the slope portal consisting of an office/shop/warehouse, supply yard,

- Each of the four mines will have a coal sizing plant at the slope portal to crush, screen, sample, and deliver coal to the overland belt system. Emergency surge piles will be provided at each mine-mouth to allow mine production to continue despite surface belt outages. These mine plants will have 1338 TPH maximum capacity (1250 TPH nominal).

- Coal haulage between the mines to the rail loadout/storage facility will be by 60- and 72-inch overland conveyor belts supported by cable suspension between concrete/steel frames on 10-foot spacing (Figure 1-9). Flexibility and easy realignment are required as the overland belt may be affected by subsidence. Transfer points will be contained in permanent structures with protection for the drive motors and dust collection to minimize fugitive dust emissions.

- The coal storage/loadout facility will consist of a barn-and-silo type storage pile of 150,000 ton capacity (two days production), with an emergency storage pile of the double-cone type. Twin reclaim conveyors would run in a tunnel beneath the two storage piles, carrying the coal up to a twin loadout structure. Train loading on either of two parallel loop tracks would be achieved by coal flowing into either of two 200-ton surge bins, then into either of two 110-ton batch bins, then through a chute into the rail cars. The ability to load two trains concurrently is believed to be necessary because of the anticipated 60,000 tons (6 trains) per day shipment level, and the three to four hour loading time required by most tariffs. Perfect sequencing of trains would be required if only one loadout was available, so a duplicate is suggested. Each of the loadout bins would have an automatic sampling system attached to provide the most representative train sample.

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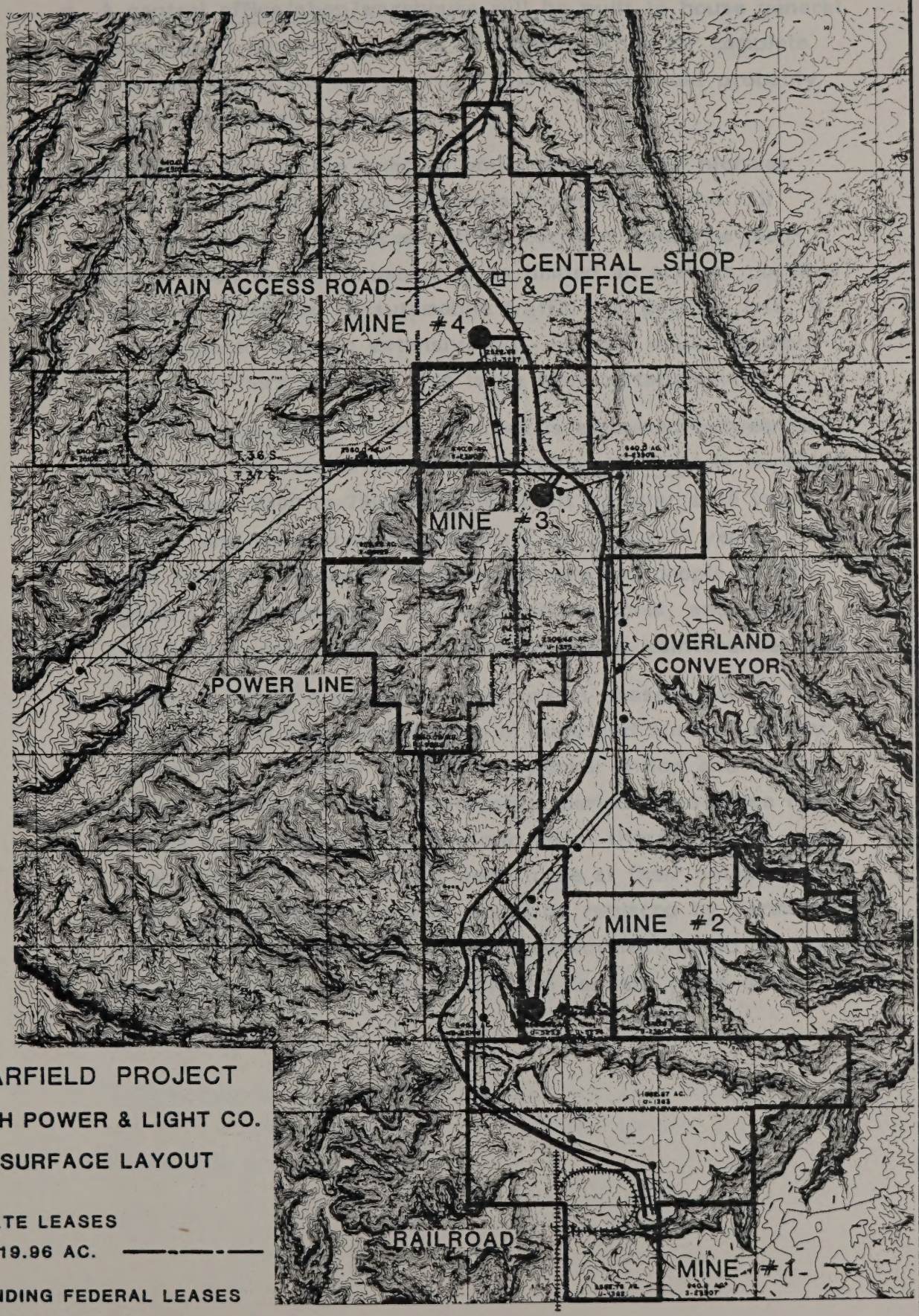


GARFIELD PROJECT

UTAH POWER & LIGHT CO.

SURFACE LAYOUT

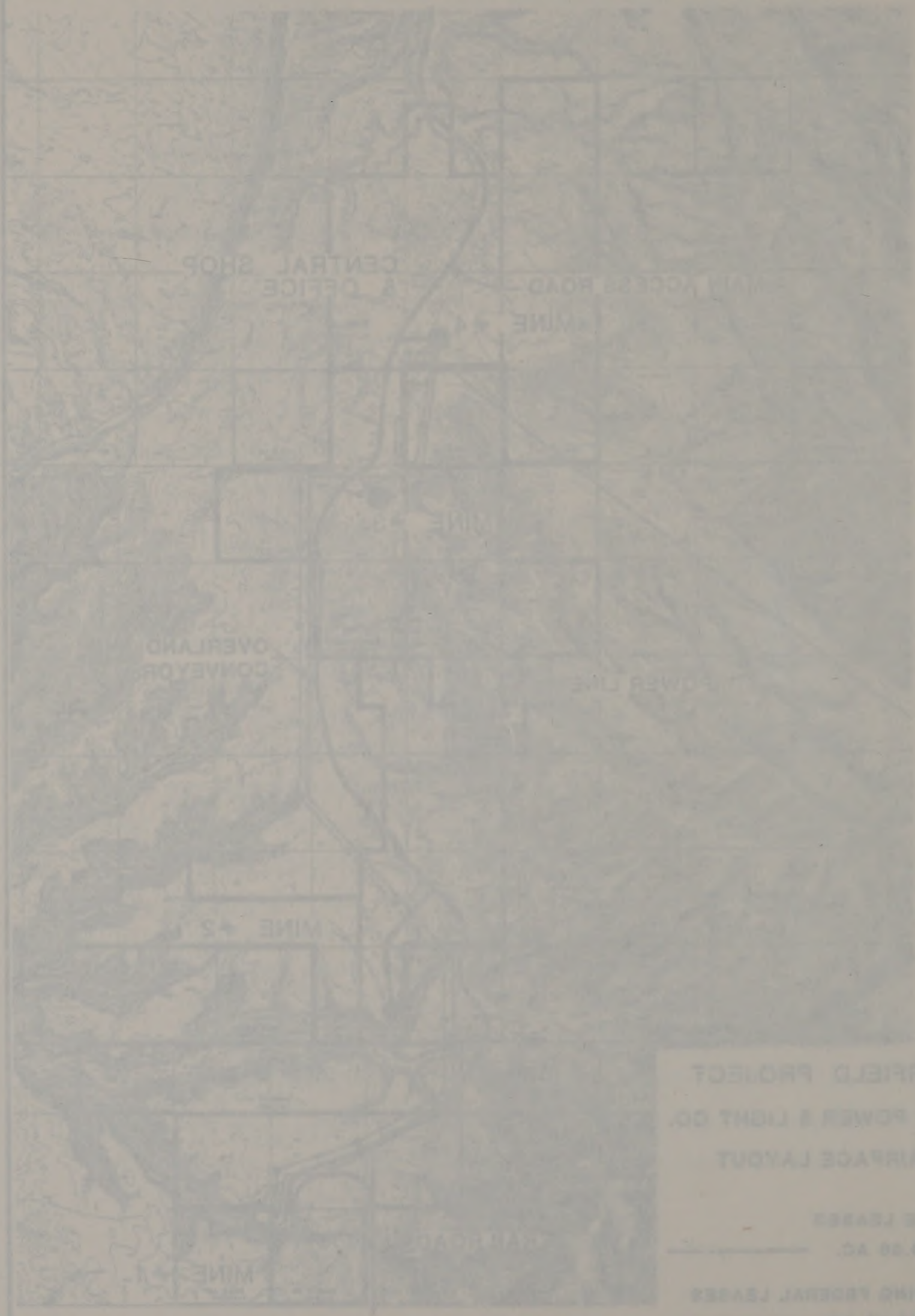
STATE LEASES

5,119.96 AC. ——-—-——

PENDING FEDERAL LEASES

18,325.16 AC. ———

FIGURE I-4



parking lot, and all utility functions, requiring about 10 to 15 acres at each mine.

- A central office/shop/warehouse will be built to house general management personnel and to provide facilities for complete overhaul of mining machinery. This facility would probably require less than 10 acres.

- Access to the mining complex would be by paved highway about 22 miles long from Escalante southward to the rail loadout facility. An access/utility corridor would be planned to keep these intrusions in as limited an area as possible. Although sized for all-automobile traffic and parking lots, it is anticipated that an extensive bus network would be developed by others to serve this large transportation need.

- Overall productivity of this complex is estimated between 25 to 28 tons per man-shift. The overall work force is expected to be about 2800 to 2900 persons, built up over a 10 year period, and maintained continuously for about 30 years. This work force is composed of about 430 salaried people in management, administration, and engineering roles, and about 2400 wage-earning people in production and maintenance roles.

- Manpower needs during the construction phase would be controlled by the sequential nature of mine and surface facility development. Surges of transient worker inflow and outflow are not expected as the construction period for surface facilities is about eight years long and all of these workers could be absorbed into the mining work force at their option. Departing construction workers would be replaced by permanent mine workers so that the maximum employment would be realized for the period between the tenth and fortieth years of the project.

- All capital and operating costs for the four-mine complex including production equipment, surface facilities, belting, access, manpower, and coal storage and loadout facilities have been determined. Capital costs of $486 MM and operating costs of $12.60 per ton are shown in Table 1-2 in the Economic Analysis Summary (Section 1.5).

- Figure 1-5 is a bargraph depicting the Overall Project Development Schedule. Predevelopment activities are shown by solid lines beginning in Year -4 and continuing for seven years. Construction activities begin in Year +1 and proceed for about nine years. Coal production is initiated in Year +3 in Mine #1, followed by sequential coal production from the other mines, shown by the diamond-shaped symbols. Railroad construction is completed and coal shipments begin during Year +4, as shown by the double diamond symbol. Continuation of mining and shipping activities for a nominal life of 33 years is depicted by the dashed lines. Declining production, dismantling, sealing, and site restoration are depicted by the dotted lines at the extreme end of the project.

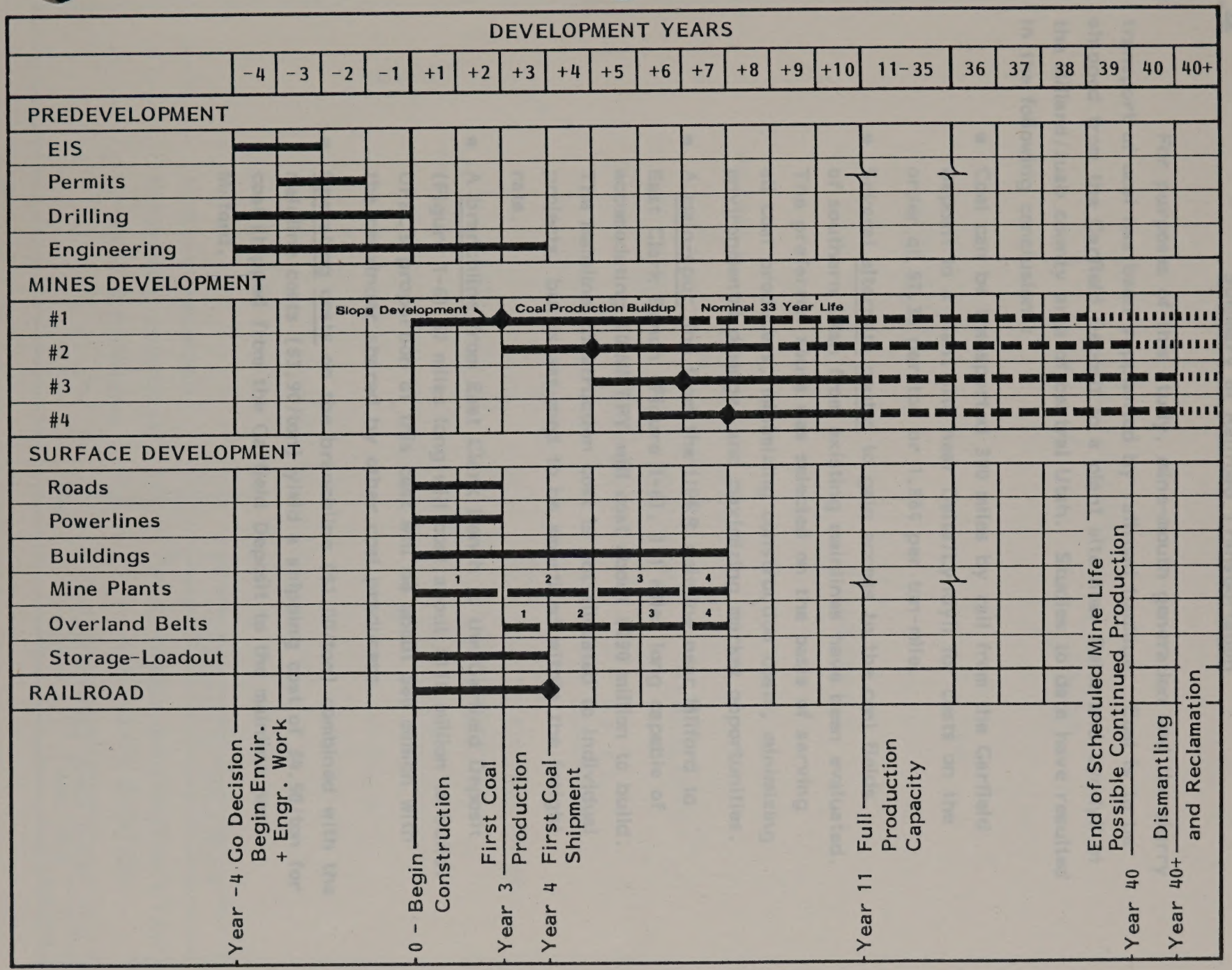
- Run-of-Mine coal quality calculations based on a 4' rock dilution for each mine face and a 1.5% moisture dilution factor results in quality projections as follows: 10,230 BTU's, 11.5% moisture, 12.6% ash, and 0.75% sulfur.

- All development plans and cost estimates have been developed in accordance with existing environmental regulations.

- All surface facilities will be removed and the ground restored to its original condition and contour. All underground openings will be sealed at the close of their productive life.

- Reclamation costs, thouth substantial, will be more than offset by the salvage value of the facilities removed during the reclamation process.

FIGURE 1-5: Overall Pr[illegible] Development Schedule

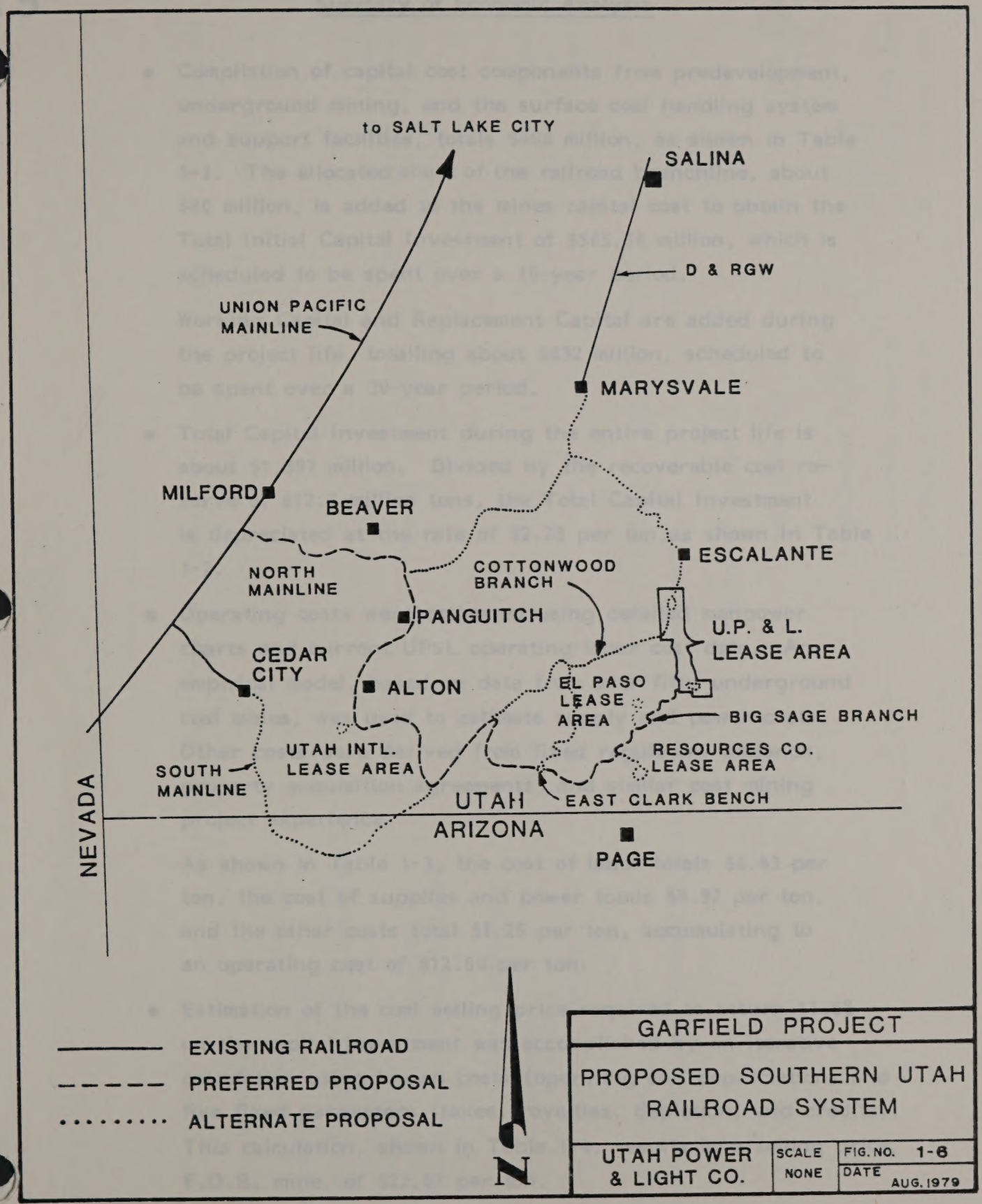


◆ Begin Coal Production

1.4 Summary of Railroad Transportation

For purposes of this study, mine-mouth generation and the slurry transport of coal has been supplanted by railroad haulage. Coal is to be shipped from the Garfield Deposit to a plant site near Delta and Lyndyll in the Millard/Juab county area of central Utah. Studies to date have resulted in the following conclusions:

- Coal can be transported 390 miles by rail from the Garfield Deposit to a plant site near Delta/Lyndyll for costs on the order of $7.35 per ton or 1.88¢ per ton-mile.
- Several alternate routes to gain access to the coal fields of southern Utah from existing mainlines have been evaluated. The preferred route was selected on the basis of serving all coal producers, minimizing construction costs, minimizing environmental damage, and maximizing market opportunities.
- A main-spur line from the UPRR mainline near Milford to East Clark Bench (Figure 1-6), 191 miles long capable of accomodating 50 MMTPY will cost about $290 million to build. The mainline construction cost is not allocated to individual projects, but is assumed to be amortized within the freight rate.
- A branchline from East Clark Bench to the Garfield Deposit (Figure 1-6) 67 miles long will cost about $110 million to build. UP&L's proportion of this cost will be about $80 million with the remainder shared by other coal producers.
- Operating costs on the branchline ($1.00/ton) combined with the mainline costs ($3.90/ton) yield a shipping cost of $4.90/ton for coal shipped from the Garfield Deposit to the mainline near Milford.


to SALT LAKE CITY
SALINA
D & RGW
UNION PACIFIC MAINLINE
MARYSVALE
MILFORD
BEAVER
ESCALANTE
NORTH MAINLINE
COTTONWOOD BRANCH
PANGUITCH
U.P. & L. LEASE AREA
CEDAR CITY
ALTON
EL PASO LEASE AREA
BIG SAGE BRANCH
UTAH INTL. LEASE AREA
RESOURCES CO. LEASE AREA
SOUTH MAINLINE
EAST CLARK BENCH
UTAH
ARIZONA
NEVADA
PAGE
EXISTING RAILROAD
PREFERRED PROPOSAL
ALTERNATE PROPOSAL
N
GARFIELD PROJECT
PROPOSED SOUTHERN UTAH RAILROAD SYSTEM
UTAH POWER & LIGHT CO.
SCALE NONE
FIG. NO. 1-6
DATE AUG. 1979

SALINA
MARYSVALE
BEAVER
MILFORD
ESCALANTE
CEDAR CITY
ALTON
PANGUITCH
PAGE
ARIZONA
UTAH
NEVADA
SOUTH MAINLINE
NORTH MAINLINE
COTTONWOOD BRANCH
U.P. & L. LEASE AREA
EL PASO LEASE AREA
UTAH INTL. LEASE AREA
RESOURCES CO. LEASE AREA
EXISTING RAILROAD
PREFERRED PROPOSAL
ALTERNATE PROPOSAL
N
BARFIELD PROJECT
PROPOSED SOUTHERN UTAH RAILROAD SYSTEM
UTAH POWER & LIGHT CO.

1.5 Summary of Economic Analysis

- Compilation of capital cost components from predevelopment, underground mining, and the surface coal handling system and support facilities, totals $486 million, as shown in Table 1-2. The allocated share of the railroad branchline, about $80 million, is added to the mines capital cost to obtain the Total Initial Capital Investment of $565.56 million, which is scheduled to be spent over a 16-year period.

  Working Capital and Replacement Capital are added during the project life, totalling about $832 million, scheduled to be spent over a 30-year period.

- Total Capital Investment during the entire project life is about $1,397 million. Divided by the recoverable coal reserve of 612.2 million tons, the Total Capital Investment is depreciated at the rate of $2.28 per ton, as shown in Table 1-2.

- Operating costs were estimated using detailed manpower charts and current UP&L operating labor cost data. An empirical model, based on data from over fifty underground coal mines, was used to estimate supply and power costs. Other costs were derived from fixed regulatory demands, property acquisition agreements, and similar cost mining project experience.

  As shown in Table 1-3, the cost of labor totals $6.43 per ton, the cost of supplies and power totals $4.92 per ton, and the other costs total $1.25 per ton, accumulating to an operating cost of $12.60 per ton.

- Estimation of the coal selling price required to return 11.6% on the capital investment was accomplished by an iterative calculation using known costs (operating and depreciation) and five fixed parameters (taxes, royalties, depletion, and credits). This calculation, shown in Table 1-4, results in a selling price, F.O.B. mine, of $22.87 per ton.

1.5 Summary of Economic Analysis

- Compilation of capital cost components from predevelopment, underground mining, and the surface coal handling system and support facilities, totals $488 million, as shown in Table 1-2. The allocated share of the railroad branchline, about $80 million, is added to the mines capital cost to obtain the Total Initial Capital Investment of $385.38 million, which is scheduled to be spent over a 10-year period.

  Working Capital and Replacement Capital are added during the project life, totalling about $822 million, scheduled to be spent over a 20-year period.

- Total Capital Investment during the entire project life is about $1,207 million. Divided by the recoverable coal reserve of 512.3 million tons, the Total Capital Investment is depreciated at the rate of $2.28 per ton,as shown in Table 1-2.

- Operating costs were estimated using detailed manpower charts and current UP&L operating labor cost data. An empirical model, based on data from over fifty underground coal mines, was used to estimate supply and power costs. Other costs were derived from fixed regulatory demands, property acquisition agreements, and similar coal mining project experience.

  As shown in Table 1-3, the cost of labor totals $6.63 per ton, the cost of supplies and power totals $4.32 per ton, and the other costs total $1.25 per ton, accumulating to an operating cost of $11.60 per ton.

- Estimation of the coal selling price required to return 11.8% on the capital investment was accomplished by an iterative calculation using known costs (operating and depreciation) and five fixed parameters (taxes, royalties, depletion, and credits). This calculation, shown in Table 1-4, results in a selling price, F.O.B. mine, of $22.27 per ton.

TABLE 1-2

SUMMARY OF CAPITAL INVESTMENT

(Millions of Dollars)

	Mine #1	Mine #2	Mine #3	Mine #4	Common	Total	Source
Predevelopment					$ 15.11	$ 15.11	
EIS, drilling, engineering, permit applications							
Underground Facilities & Equipment							
Seam Access	$ 9.78	$ 17.39	$ 16.35	$13.83		57.35	Table 3-9
Production Units	35.61	59.38	59.38	59.38	38.96*	252.71	Table 3-12
Belts, Haulage, Utilities	15.54	20.95	20.72	19.81		77.02	Table 3-17
*spare equipment not assigned to a mine						$ 387.08	
Surface Facilities & Equipment							
Mine Buildings	4.38	4.47	4.47	4.49	6.27	$ 24.08	Table 3-18
Access & Utilities					18.16	18.16	Table 3-19
Coal Handling System	1.40	2.00	2.00	2.00	34.21	41.61	Table 3-20
						$ 83.85	
Mine Totals	$66.71	$104.19	$102.92	$99.51	$112.71	$486.04	
Railroad (Big Sage Branch, Prorated share of capital costs)					$ 79.52	$ 79.52	
Total Initial Capital Investment						$565.56	
Working Capital						-	($30.0 MM retrieved at end of project)
Replacement Capital						$ 831.66	
						$1397.22	
Recoverable Reserve						612.22	MM Tons
Depreciation Charge						$ 2.28	per ton

- Addition of the rail freight tariff rate for transporting coal from the Garfield Project to the proposed generating station near Delta-Lyndyll, Utah is also shown in Table 1-4. This freight rate, $7.35 per ton, increases the delivered cost of Garfield coal to $30.22 per ton, or 148¢ per MMBTU.

TABLE 1-3

MINE OPERATING COST SUMMARY

GARFIELD PROJECT

Salaried Employees	$ 0.85 per ton
Hourly Employees	4.20
UMWA Pension Fund	1.38
Supplies and Materials	4.31
Power	0.61
Insurance, Taxes, Royalties	0.60
Federal Funds (OSM, MSHA)	0.65
	$12.60 per ton

TABLE 1-4

FINANCIAL ANALYSIS SUMMARY

GARFIELD PROJECT

Known Costs

Operating Cost, from Table 1-3	$12.60 per ton
Depreciation, from Table 1-2	2.28

Fixed Parameters

Federal Royalty = 8% of Selling Price
State Income Tax = 4% of Net Income
Federal Depletion = 10% of Selling Price less Royalty
Federal Income Tax = 46% Net Income less State Tax and Depletion
Investment Tax Credit = 10% of Depreciation

F.O.B. Mine Selling Price Determination

(Final trial-and-error iteration is shown)

Sale Price	$22.87
Royalty	1.83
Operating Cost	12.60
Depreciation	2.28
Net Before Taxes	6.16
State Income Tax	.25
Net Before Depletion	5.91
Federal Tax Depletion Allowance	2.10
Net Taxable for Federal Purposes	3.81
Federal Income Tax	1.75
Investment Tax Credit	.23
Net after Taxes	2.28
Depletion Addback	2.10
Depreciation Addback	2.28
Net Cash Inflow from Operations	$ 6.67

Rail Transportation Costs

Operating Cost, 67 miles Big Sage Branch	$ 1.00 per ton
Total Cost, 191 miles, East Clark Bench to Milford	3.90
Total Cost, 132 miles, Milford to Delta/Lyndyll	2.45
	$7.35 per ton

Delivered Price of Garfield Project Coal to Proposed IPP-ICPA Plant

Coal Price, F.O.B. Mine	$22.87 per ton
Rail Tariff, Mine to Delta	7.35
	$30.22 per ton

÷ 10,230 BTU/lb = 148¢ per MMBTU

Acknowledgements

This Resource Development Evaluation is the collective effort of seven organizations. Project management and mine planning services were provided by Skelly and Loy, Consulting Engineers, Denver, Colorado, and Harrisburg, Pennsylvania.

Input in specialized areas was provided by Mr. John Peperakis, Mining Consultant, Salt Lake City; Coal Systems, Inc., Salt Lake City; and NUS Corporation, Energy Consultants, Denver, Colorado.

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The principal authors of this report were:

Roger S. Dewey, P.E., Skelly and Loy, Denver
Gerald E. Vaninetti, Utah Power & Light Company

Major contributors included:

Skelly and Loy ---- W. Ernest Hilton, John T. Brown, and Don Polly,
Coal Systems, Inc. ---- L. G. Manwaring, P.E., and Marilyn Millner
NUS Corporation ---- William F. Jennings, P.E.
UP&L Mining & Exploration Department ---- Sam Johnson, and the Services and Secretarial Groups.

Log of Photographs

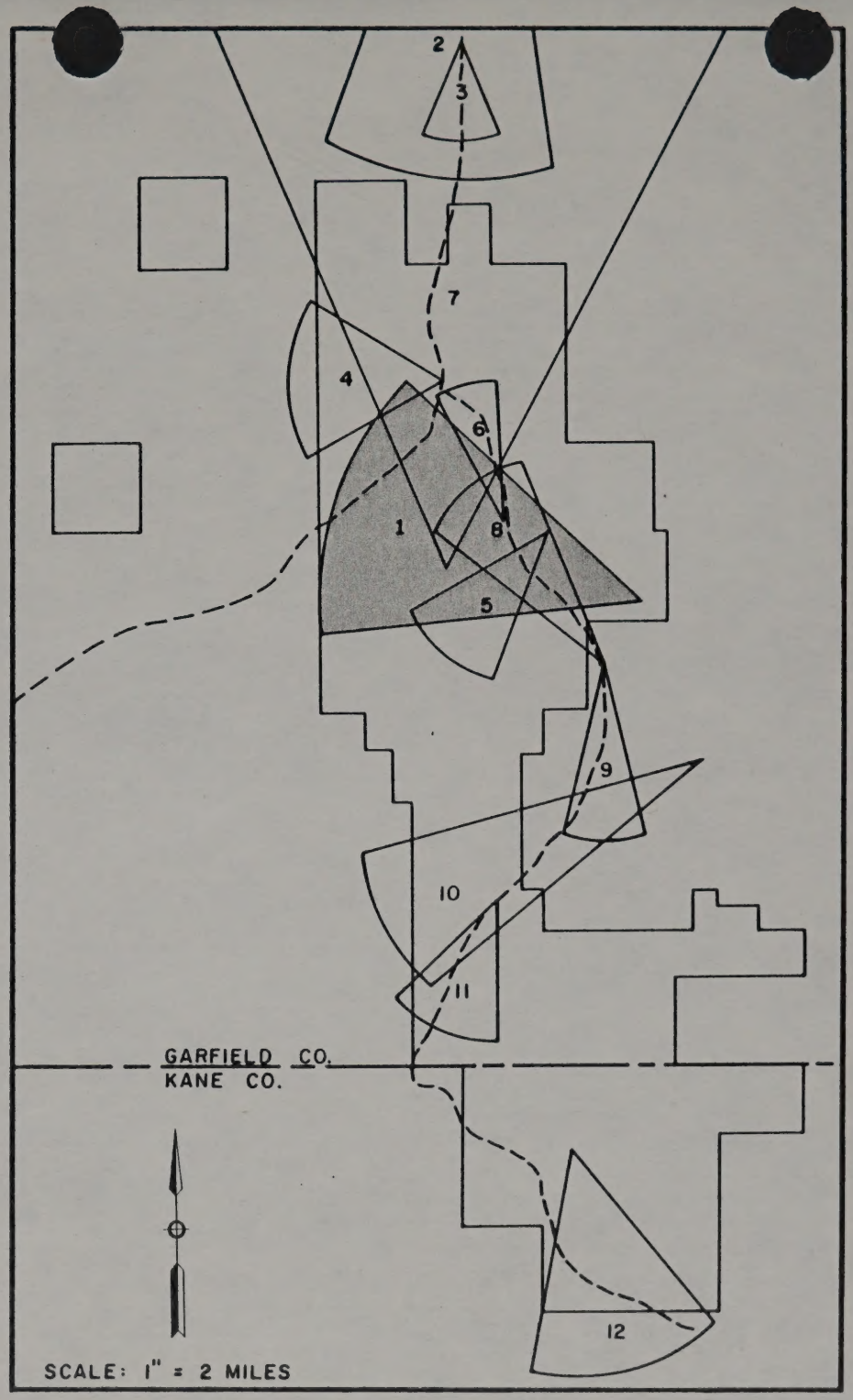
A group of twelve photographs have been compiled to illustrate the physiography, geology, access, topography, and vegetation of the Garfield Deposit. Both aerial and ground photographs have been sequentially organized to permit a general visual examination of the region adjacent to and on the deposit progressing from the north, near Escalante, to the south, at the southern end of the deposit. A general outline of the deposit, locations of viewpoints, and a photo description are presented on the pages that face the photographs in order to orient the viewer.

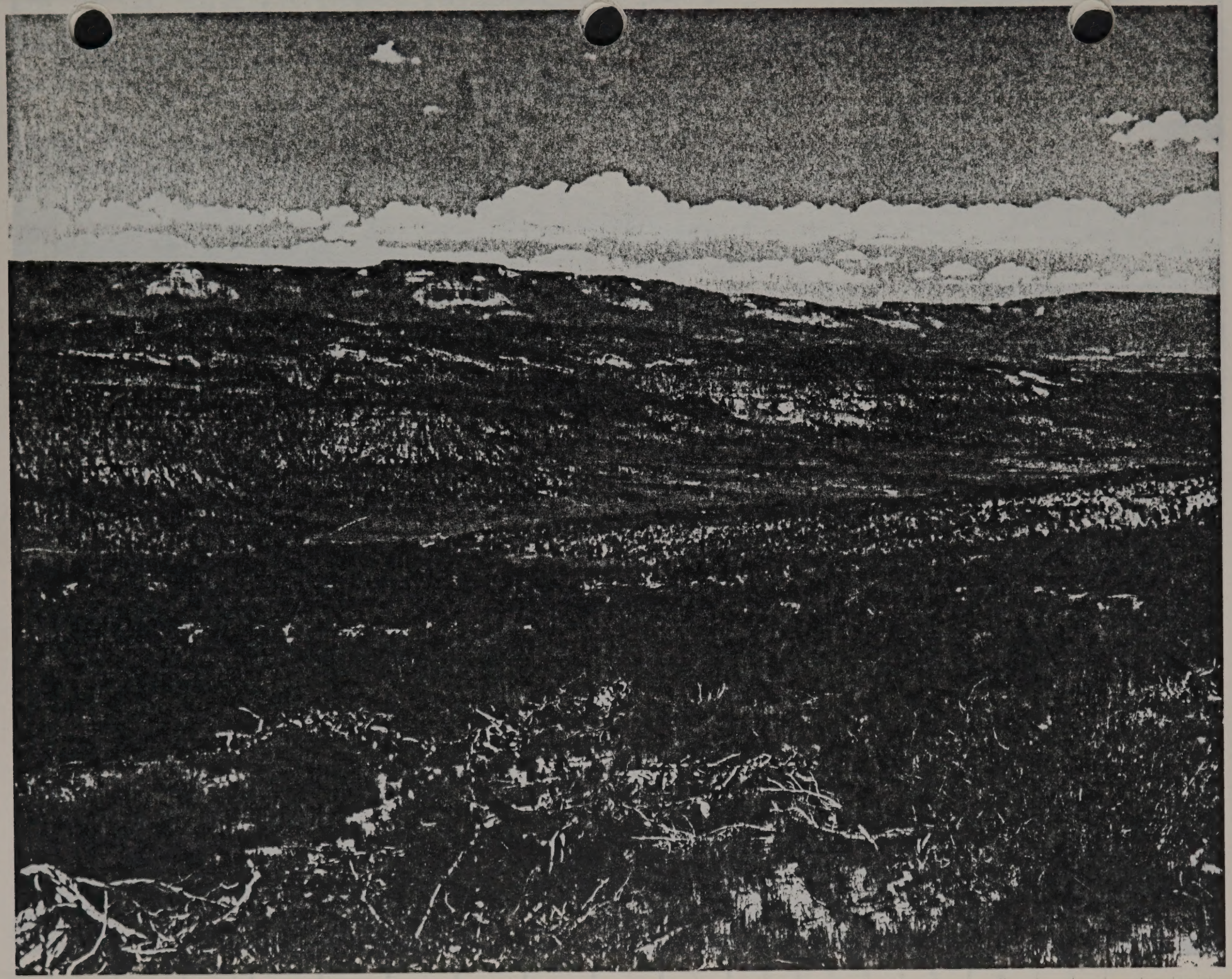
PLATE 1-1.

View to the Northwest into Little Valley from north end of Camp Flat.

#3 Mine site at left center.

#4 Mine site at right center.





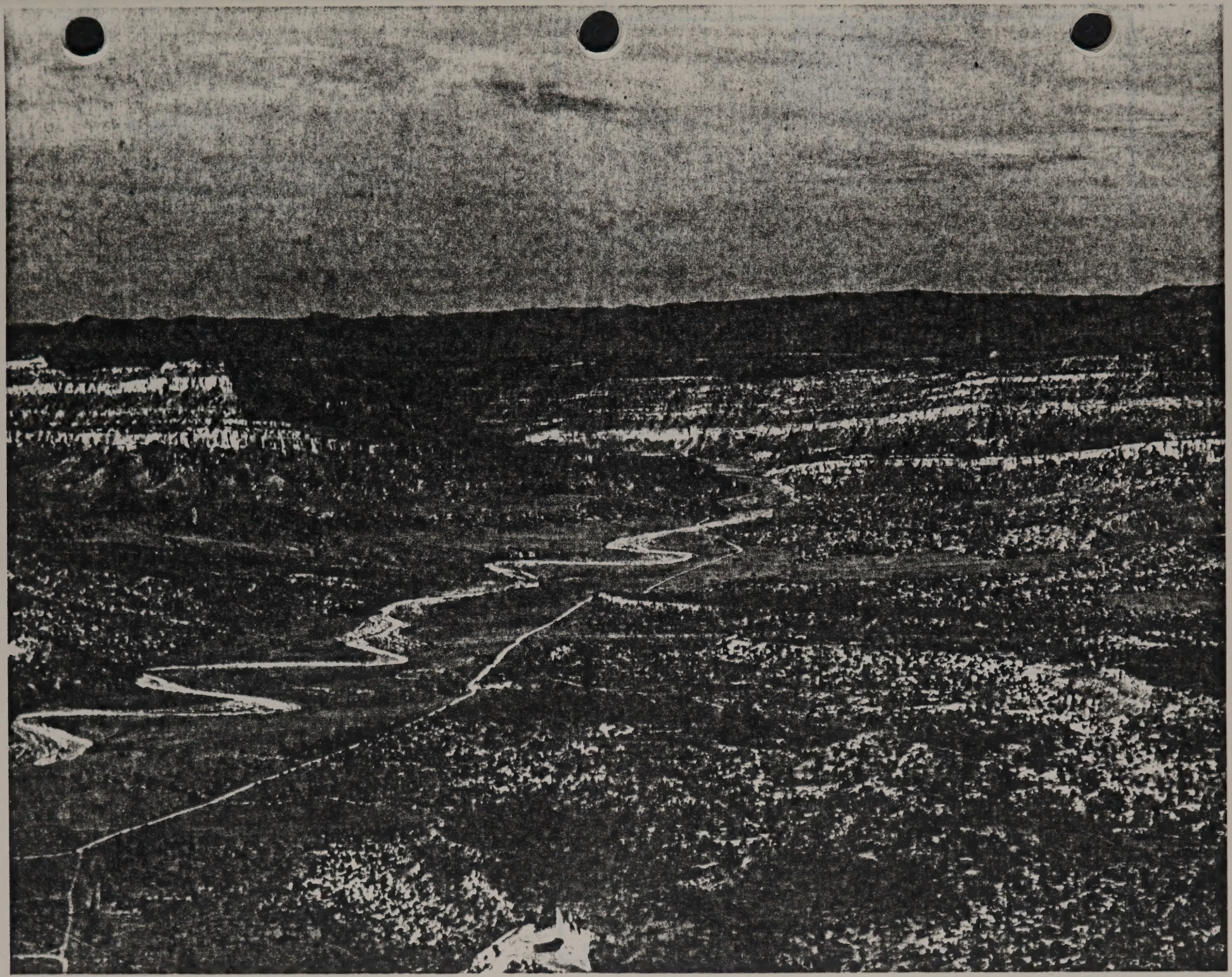
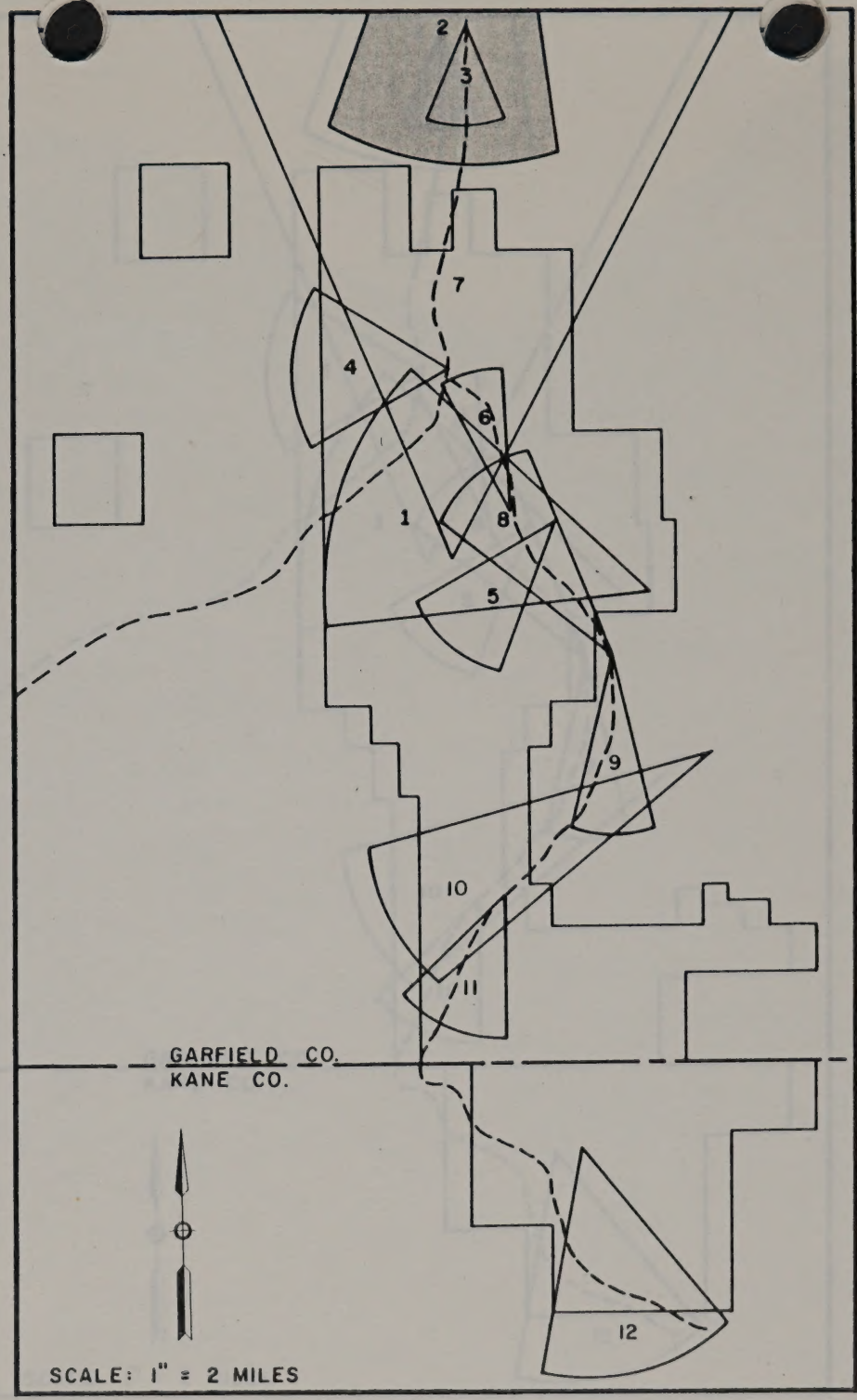


PLATE 1-2.

View to Southwest into mouth of Alvey Wash from vicinity of town of Escalante.

Main access road would follow the existing road which parallels the dry wash.

Coal seams outcrop above massive sandstone beds.



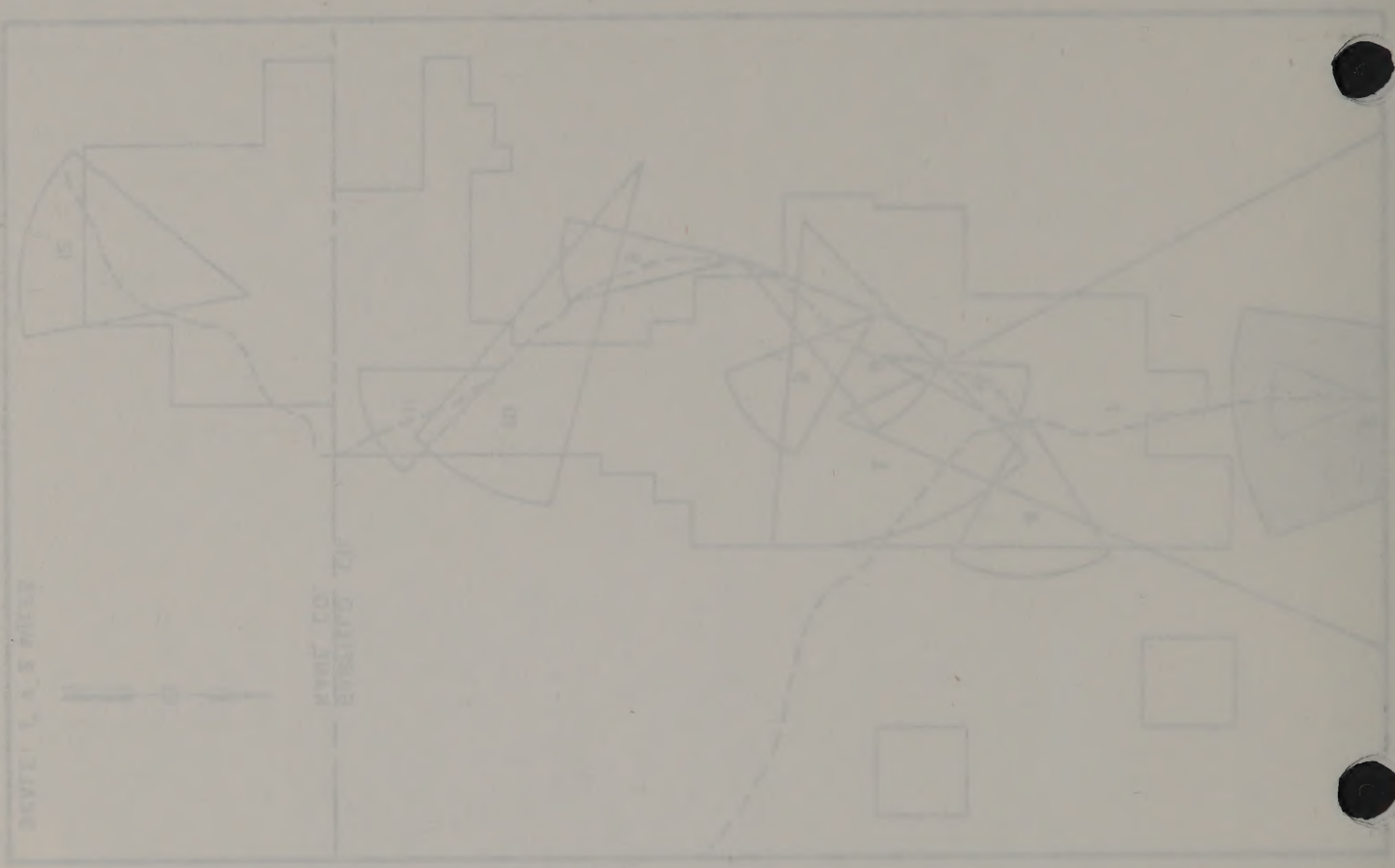
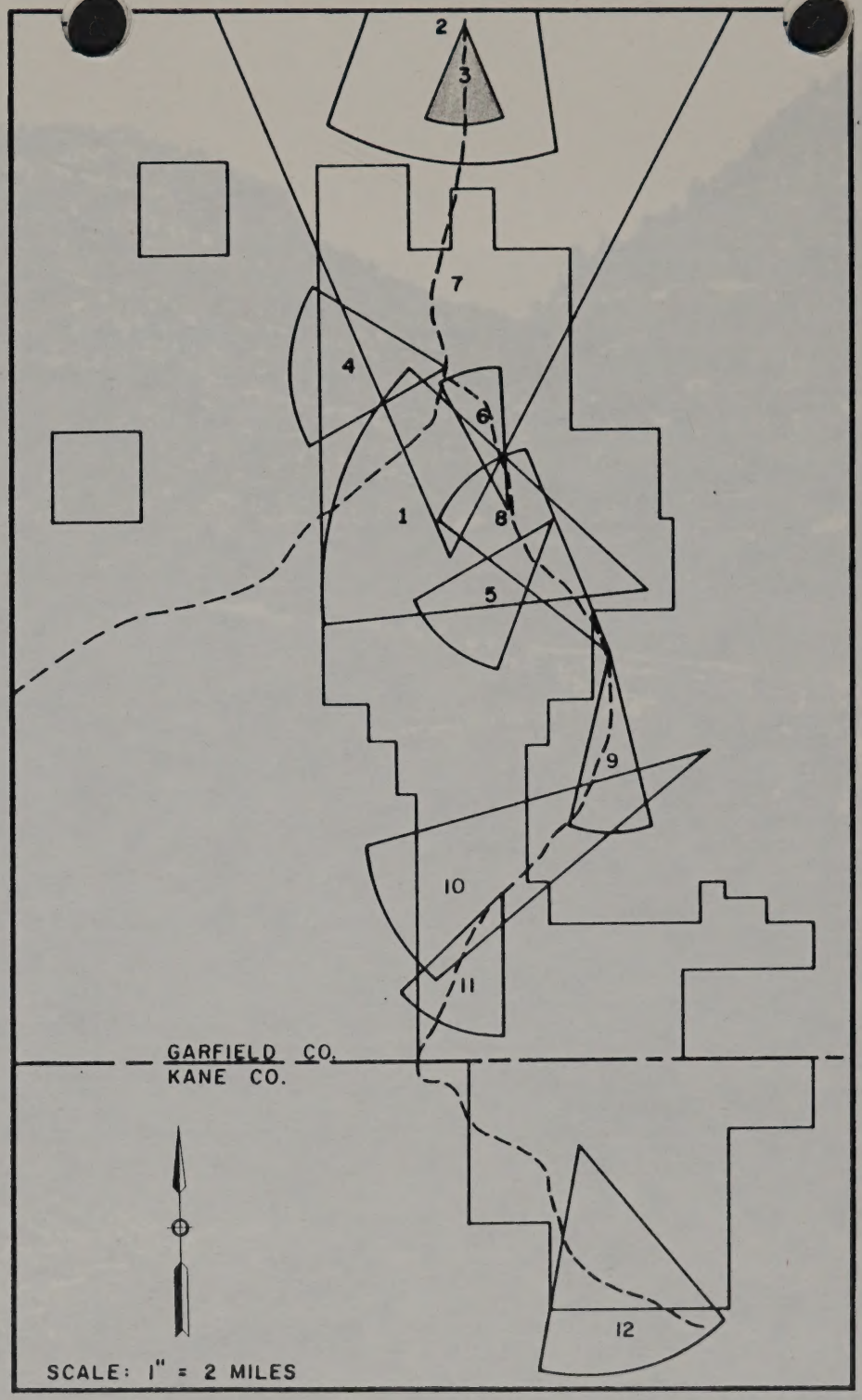


PLATE 1-3.

View to South of Alvey Wash.



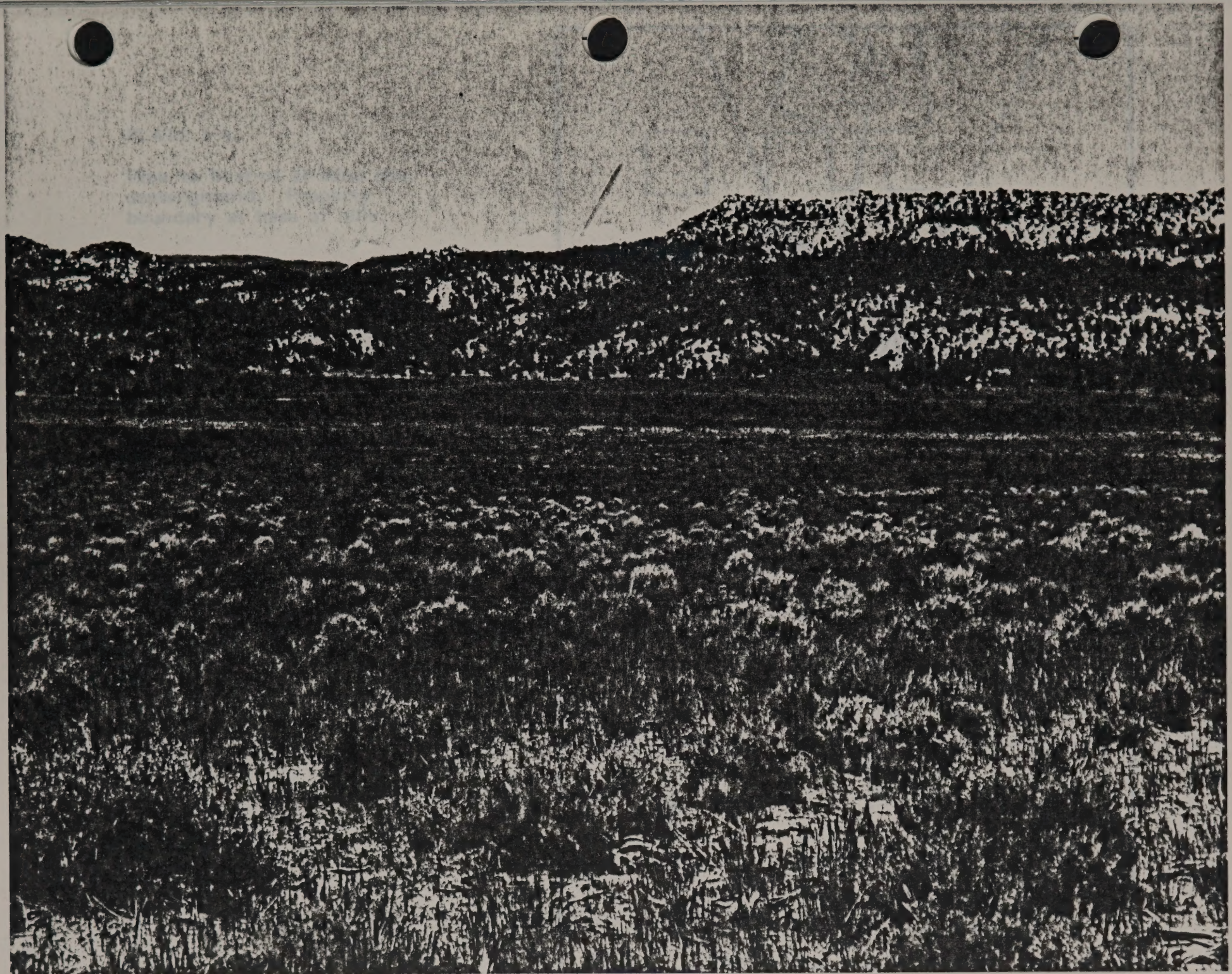


PLATE 1-4.

View to West of #4 Mine site, centerground. Property boundary at base of hills.

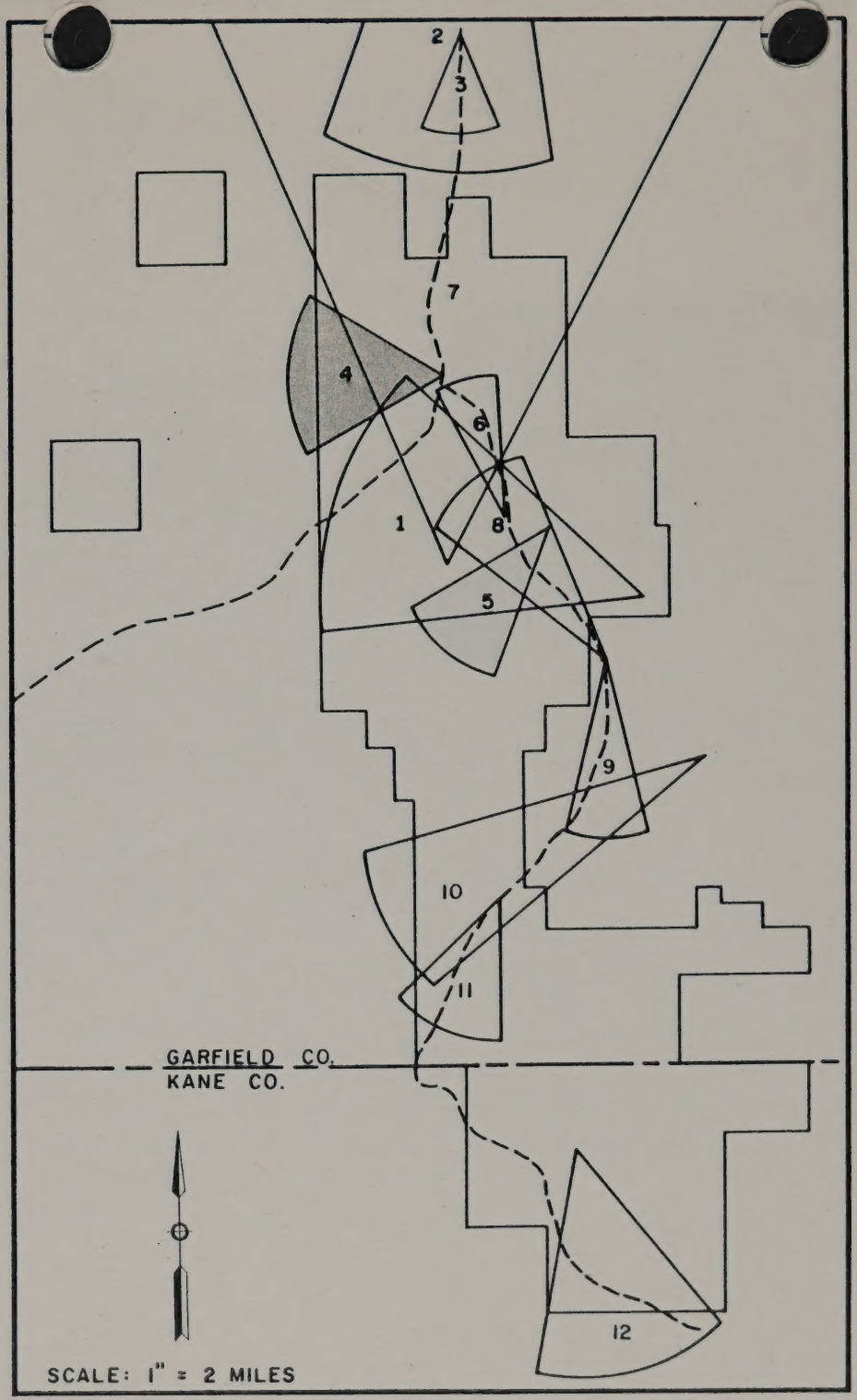


PLATE 1-5.

View to South of #3 Mine site centerground.

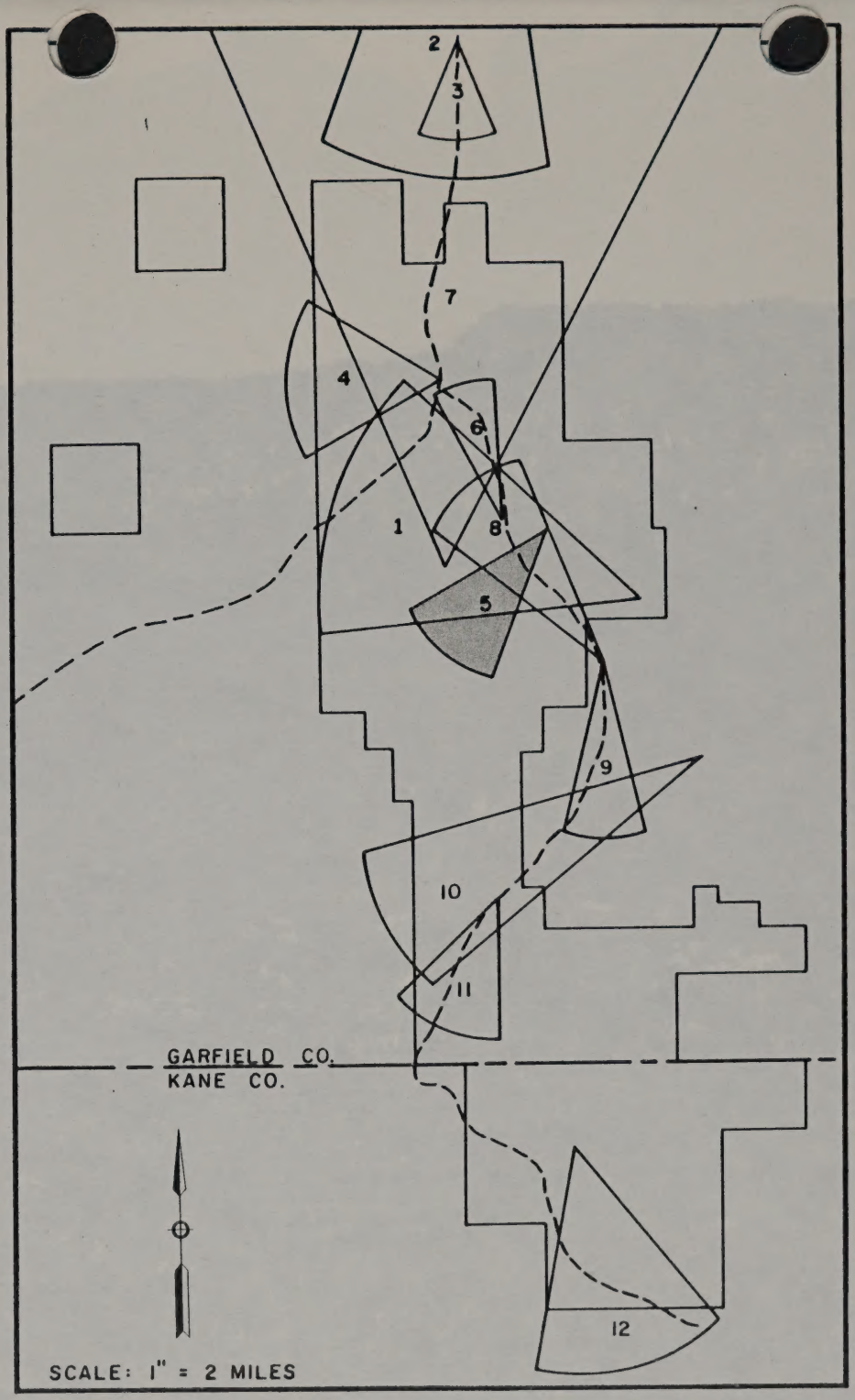


PLATE 1-6.

Access road and overland belt route between #4 Mine and #3 Mine. View to Northwest.

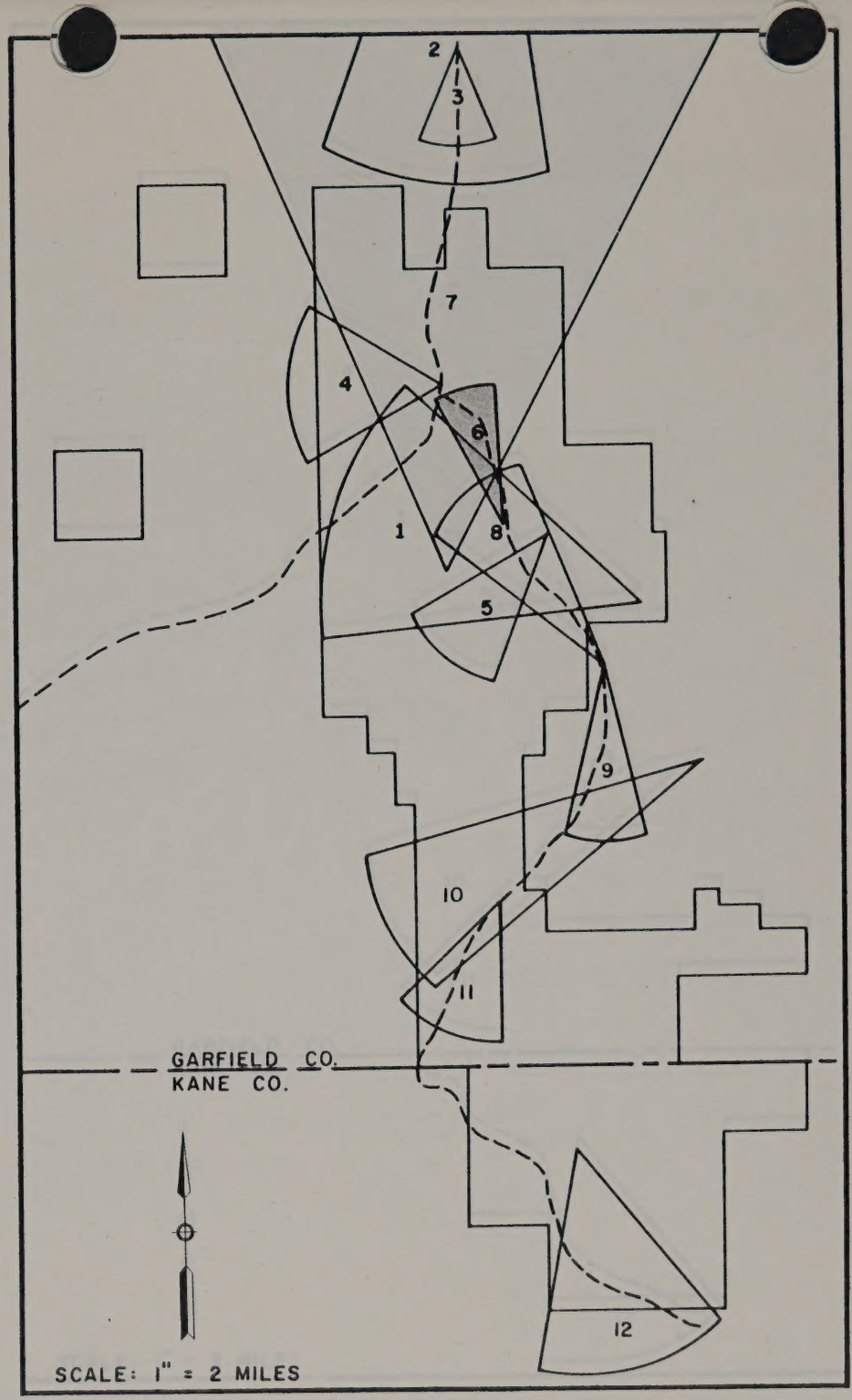
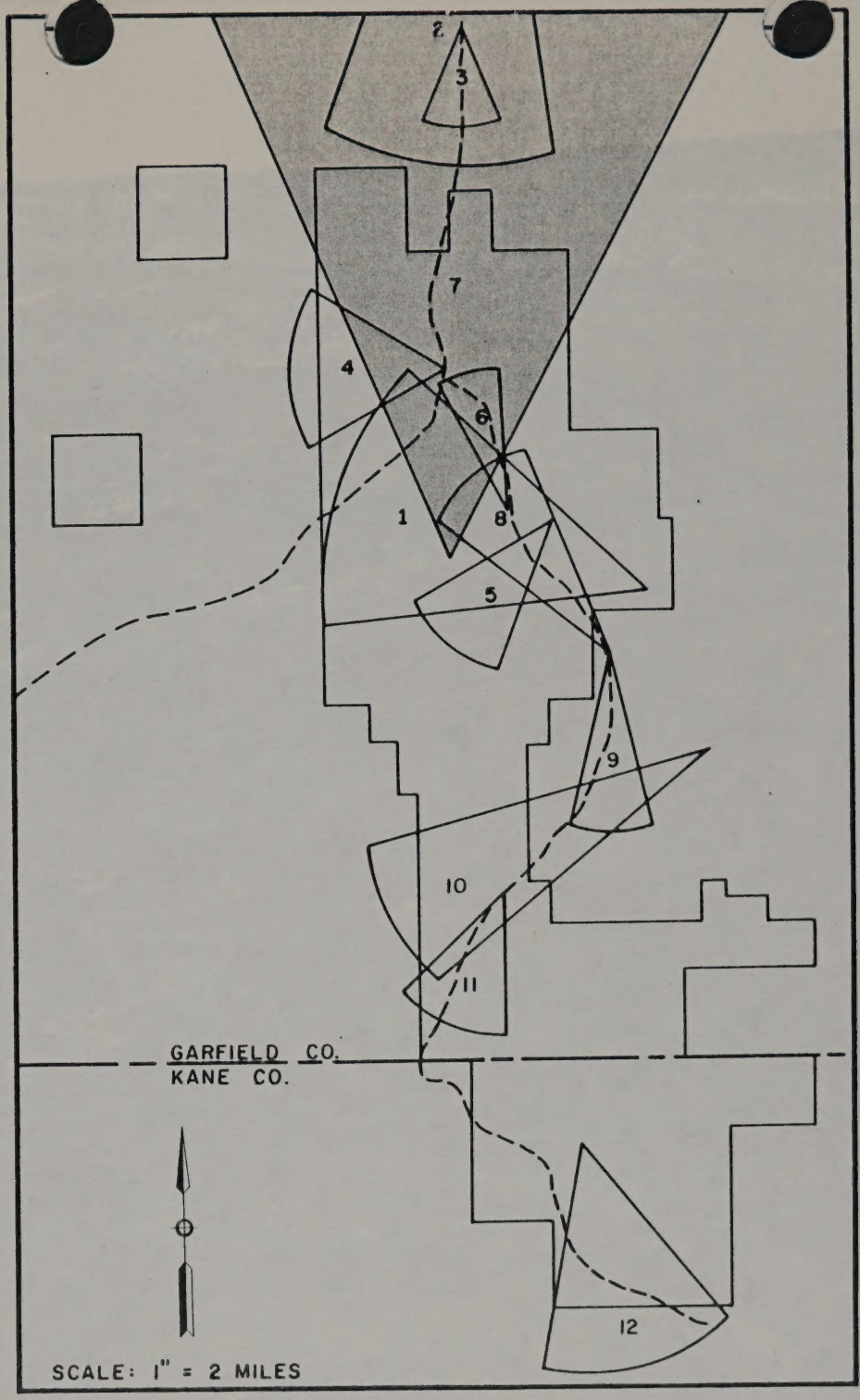


PLATE 1-7.

Aerial view of Little Valley and Alvey Wash, looking North. #4 Mine site at left center along road. Photo 1-6 taken on road in right foreground. Town of Escalante in upper right.



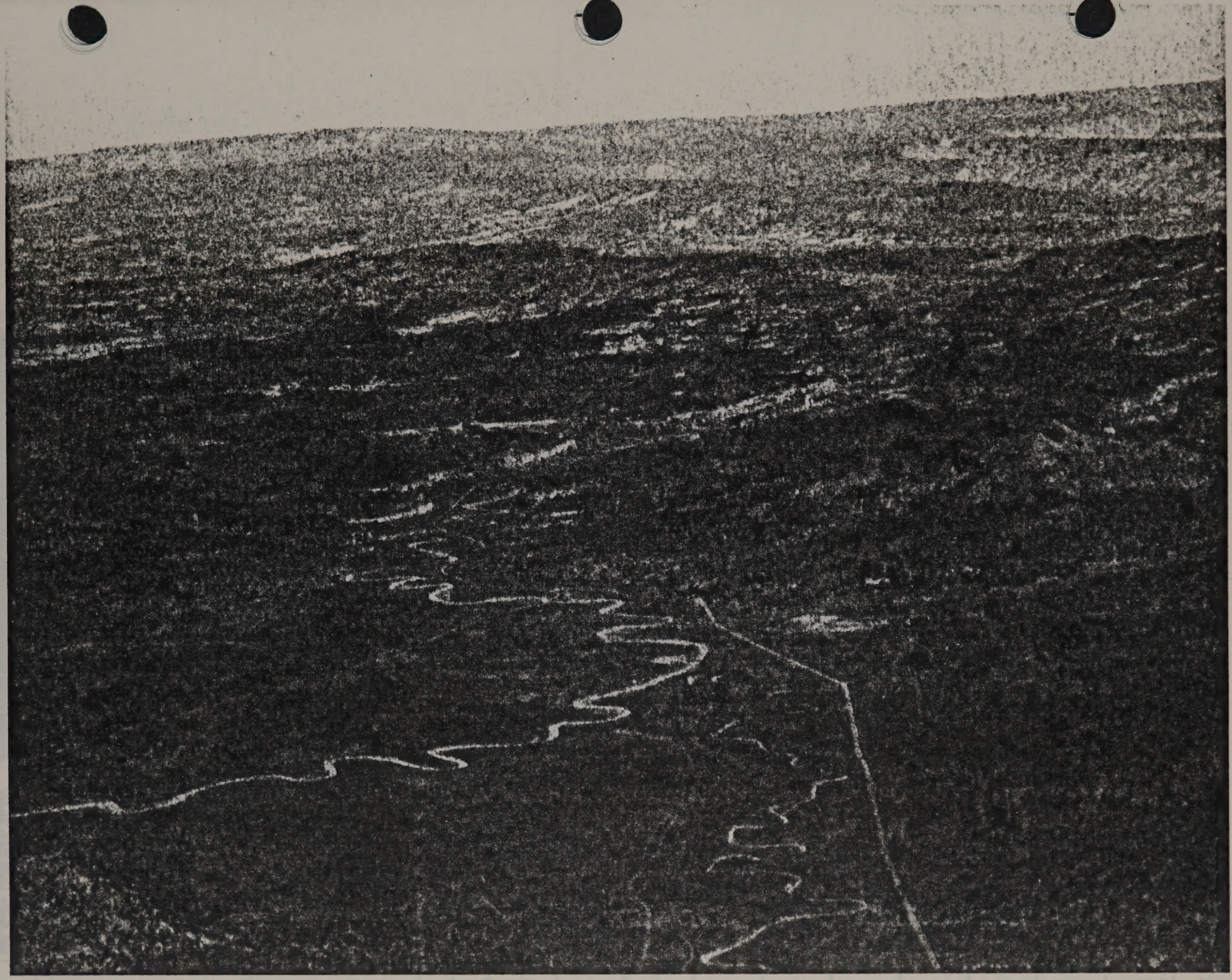


PLATE 1-E

View of access road and overland belt route between Little Valley and Camp Flat. View to Northwest. #3 Mine site in left center and #4 Mine site in center of photo.

PLATE 1-8.

View of access road and overland belt route between Little Valley and Camp Flat. View to Northwest. #3 Mine site in left center and #4 Mine site in center of photo.

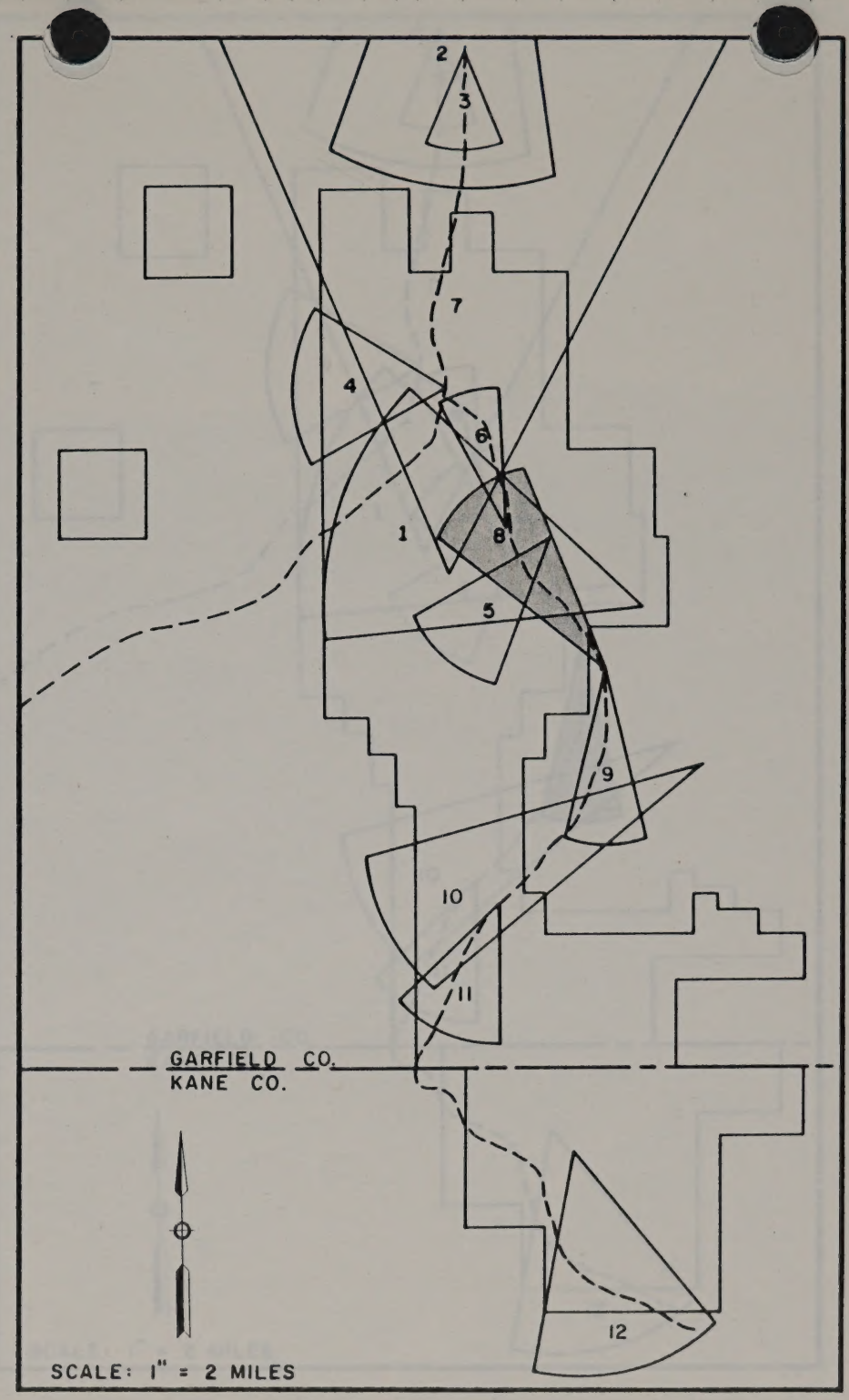
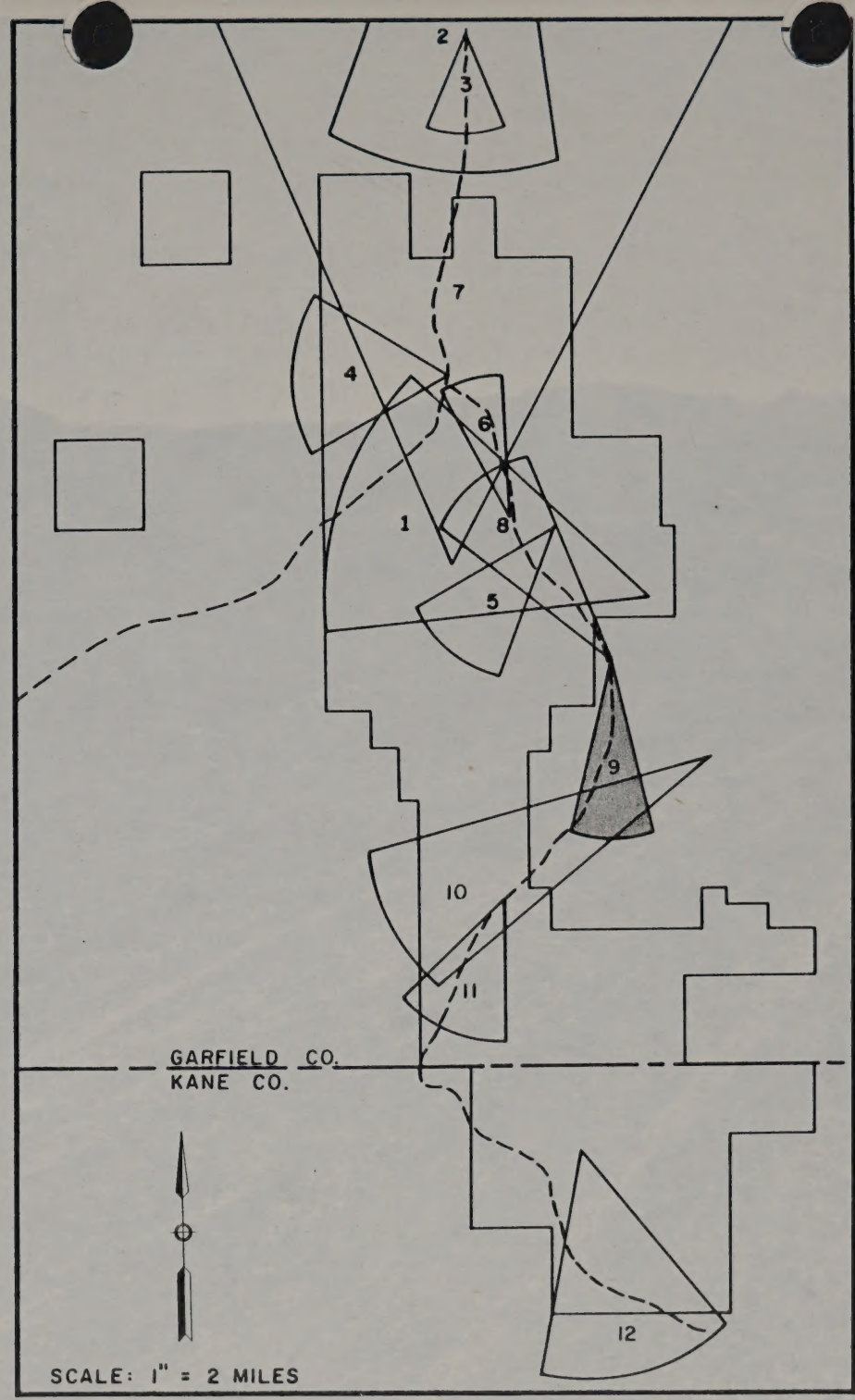
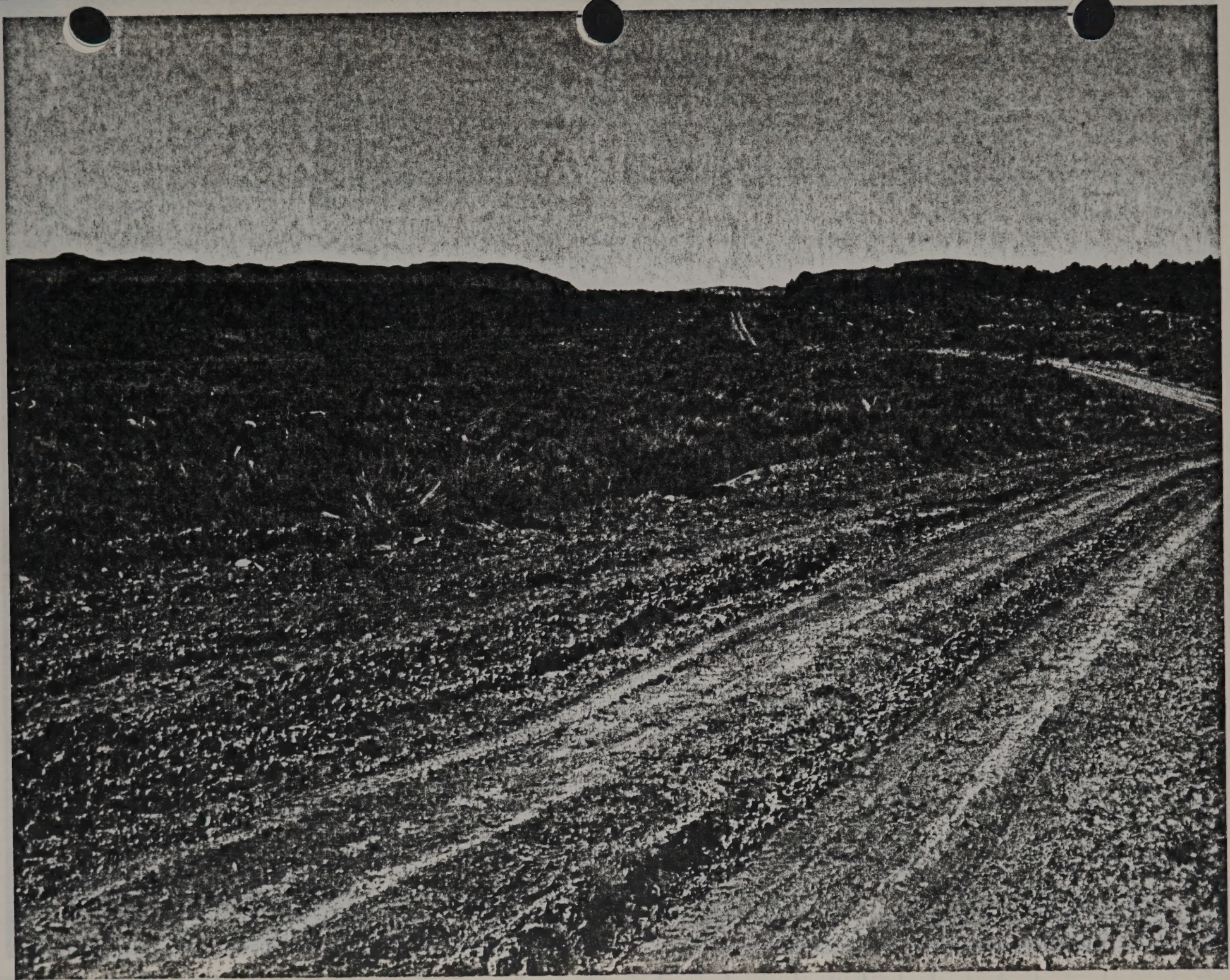


PLATE 1-9.

Camp Flat area, access road and overland belt route. View to South.





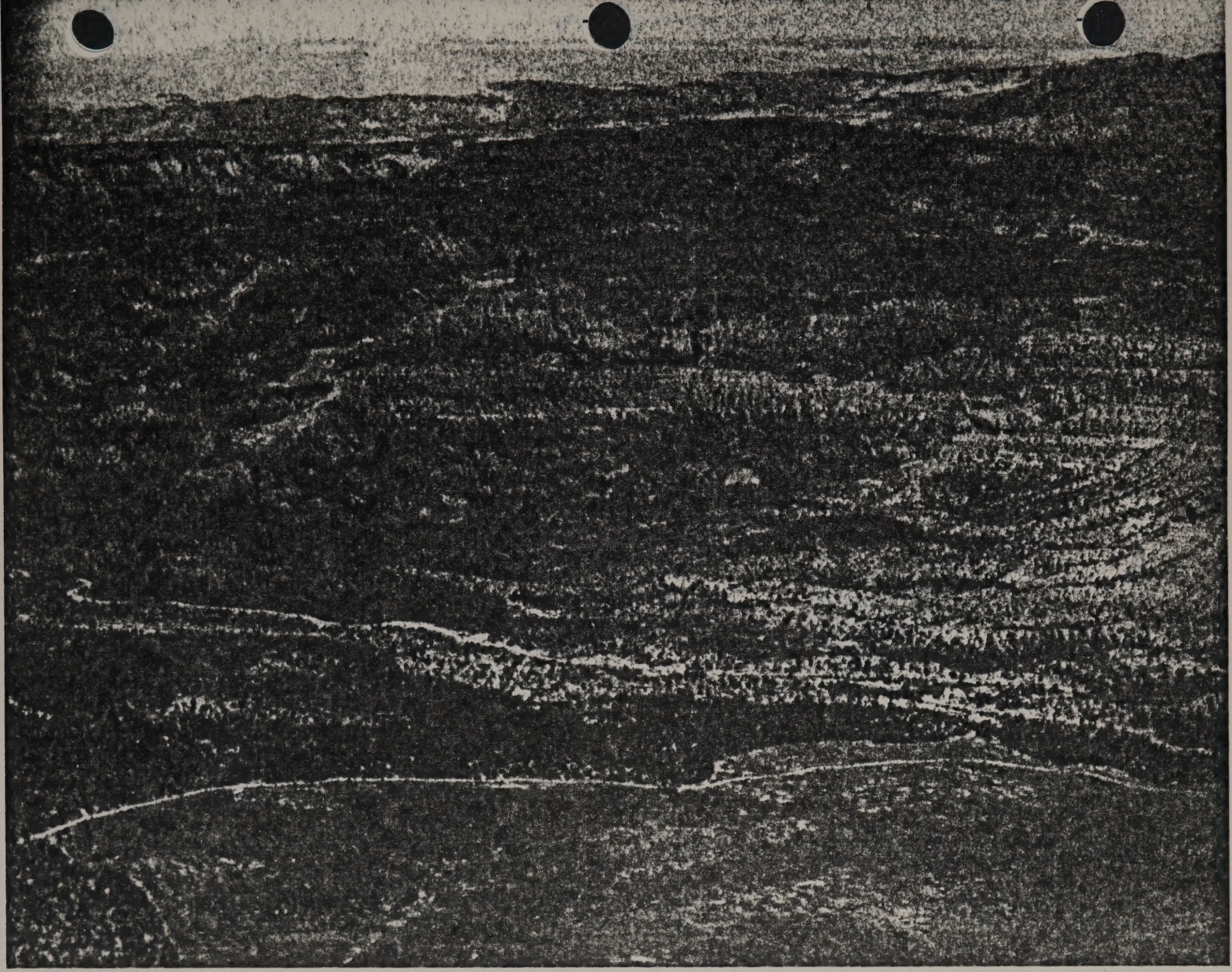


PLATE 1-10.

Aerial view of Camp Flat (south end), looking Southwest. Upper road is on route of access road and overland belt.

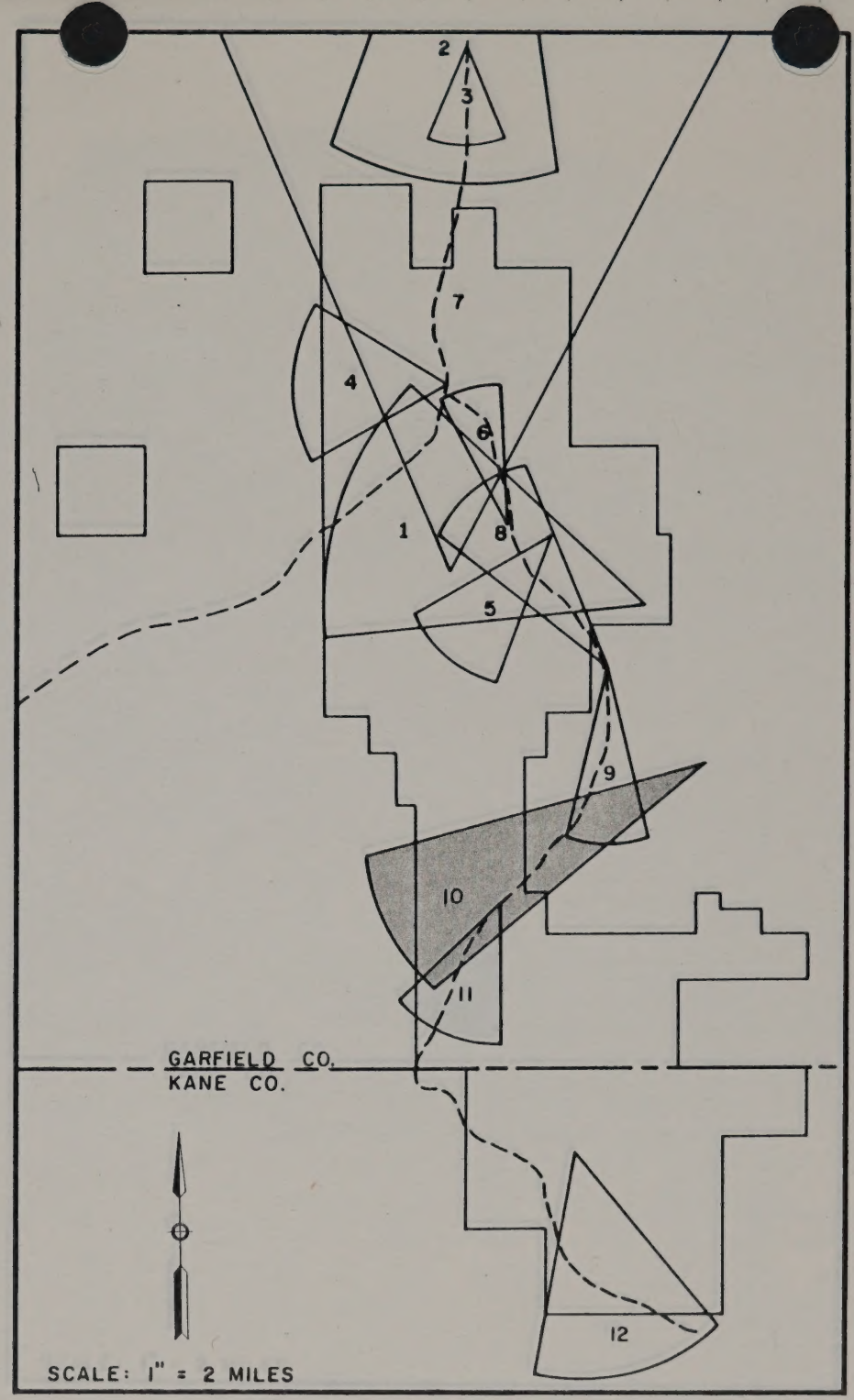


PLATE 1-11.

Camp Spring area, Northwest branch of Right Hand Collet Canyon in middleground, looking Southwest. Transfer point of belt from #2 Mine to be located in foreground. Access road and overland belt to run southward on bench in middleground.

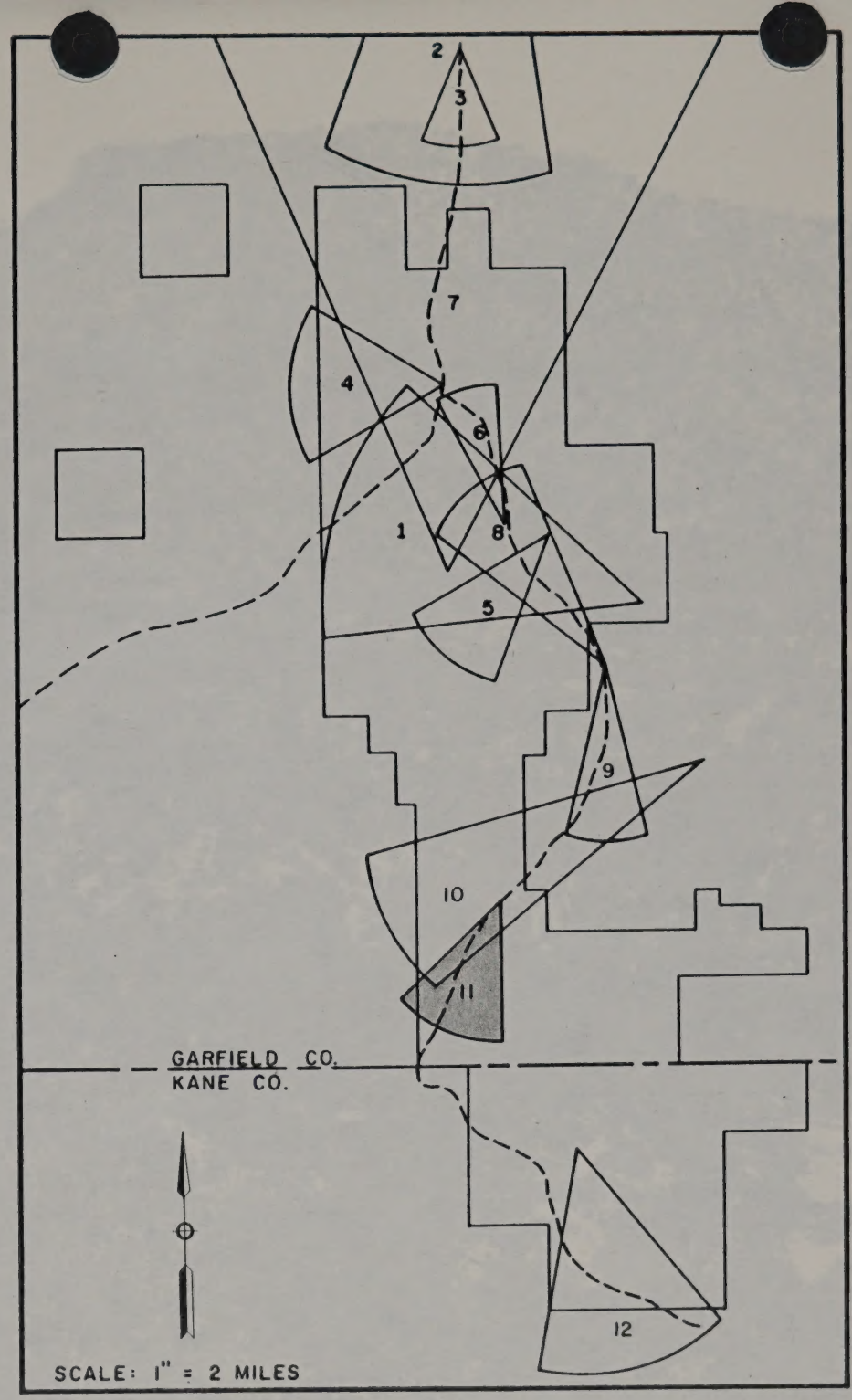
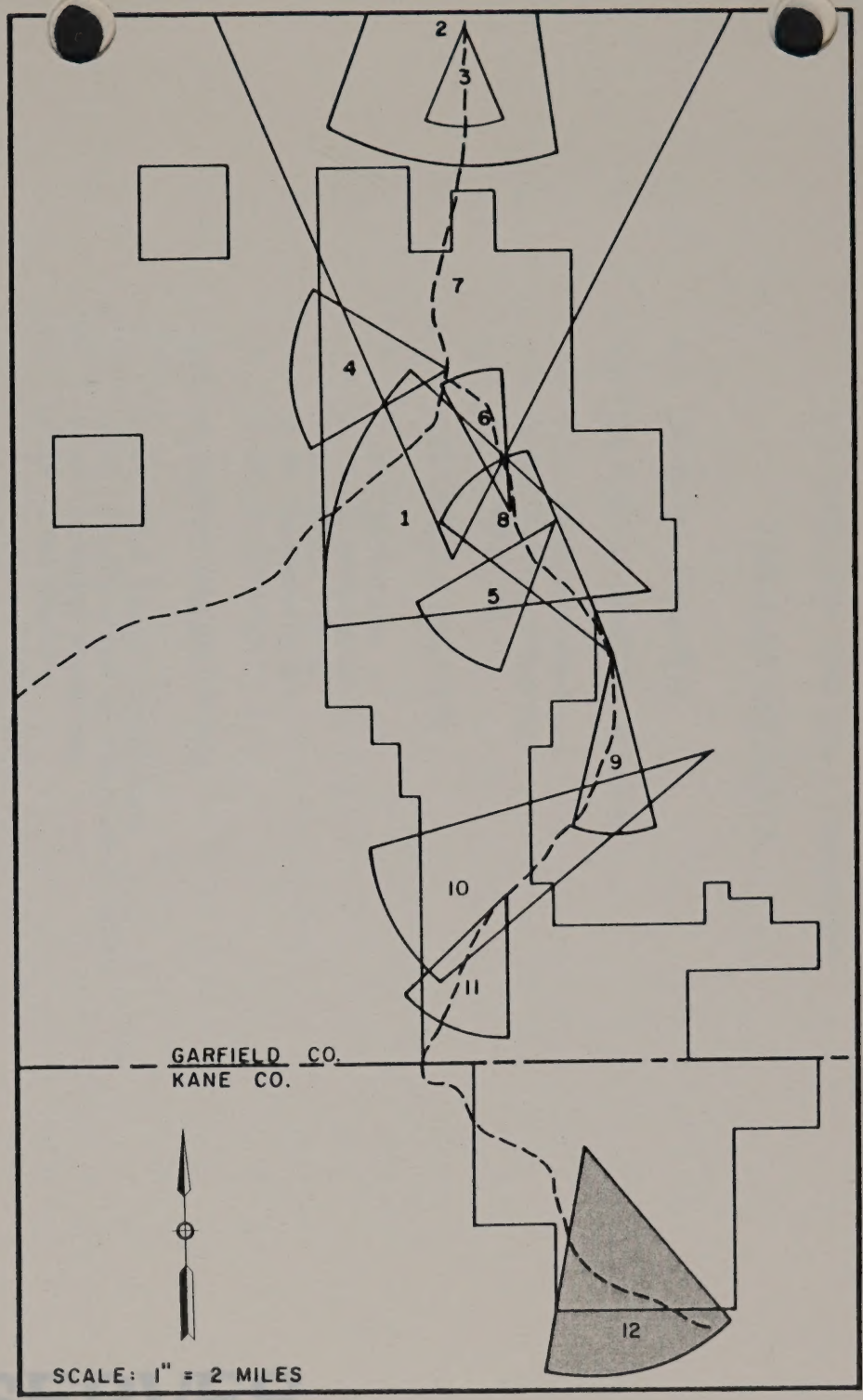


PLATE 1-12.

Little Sage Flat, looking South. The #1 Mine site and the coal storage - loadout - railspur loop facility to be located here. The dead brush remains from a 1960's churning program.



2 GEOLOGY & RESERVES

2 GEOLOGY & RESERVES

U.S. GEOLOGY & RESERVES

2.0 GEOLOGY AND COAL RESERVES

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2.0 GEOLOGY AND COAL RESERVES

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2. Geology and Coal Reserves

General Geology 2.1

The coal-bearing sedimentary rocks of the Kaiparowits Plateau were first investigated by Gregory and Moore as reported in U.S.G.S. Professional Paper 164 (Gregory and Moore, 1931). The stratigraphic nomenclature that they developed is still used, with minor modification, throughout the region. Subsequent U.S.G.S. workers (notably Peterson, Zeller, Waldrop, and Bowers) have mapped many of the seven-and-one-half minute quadrangles within the plateau and have shed additional light on the stratigraphic and sedimentologic character of the sedimentary units within the region.

The mapping conducted by the U.S.G.S. workers noted above indicated that the major coal reserves of the region are included within the Straight Cliffs Formation (Figure 2-1). This formation is overlain by the Wahweap Formation and overlies the Tropic Shale. The coal-bearing portion of the Straight Cliffs Formation is age equivalent to the coal-bearing unit at Black Mesa, Arizona and the Blue Gate Shale and Emery Sandstone Members of the Mancos Shale in the Wasatch Plateau of central Utah (Figure 2-2). The Tropic Shale is equivalent in age to the Tununk Shale Member of the Mancos Shale in the Henry Mountains and Wasatch Plateau of Utah. The Wahweap Formation is correlative to the Star Point and Castlegate Sandstones and the Blackhawk and Price River Formations in the Wasatch Plateau of Utah. Both the Straight Cliffs and Wahweap Formations are extensively exposed on the Garfield Deposit.

The Kaiparowits Plateau is characterized, in a structural sense, by a series of north-northwest trending anticlines and synclines that plunge at a low angle to the north (Figure 2-3). For the most part, these folds are gentle with dips on the flanks of structures rarely exceeding 10°. The Garfield Deposit is nearly centered on the Alvey Syncline and dips for the most part are less than 5° into the syncline although a reading of 10° has been encountered in the easternmost part of the deposit.

2. Geology and Coal Reserves

2.1 General Geology

SYSTEM	SERIES	STRATIGRAPHIC UNIT	THICKNESS (Feet)	DESCRIPTION
CRETACEOUS		Kaiparowits Formation	2,000-2,5000	Gray to dark gray, fine- to moderately coarse-grained, friable 'salt and pepper' arkosic sandstone with subordinate light gray mudstone; weak calcareous cement, forms badlands and slopes.
		Wahweap Formation	760-1,350	Yellow-gray resistant sandstone, gritstone and conglomerate alternating with yellow-orange nonresistant sandstone and gray mudstone; lower half dominantly nonresistant, upper half massive and hard.
		Straight Cliffs Formation, Upper, Drip Tank Member	100-350	Yellow-brown to gray-orange, fine- to medium-grained sandstone with some gritstone and conglomerate interbedded with subordinate gray shale; resistant cliff former.
	Santonian	Straight Cliffs Formation, Upper, John Henry Member *Major Coal Seams*	500-900	Interbedded yellow-gray, white and orange medium-grained sandstone, gray shale, carbonaceous mudstone and coal; forms ledgy outcrops; often exhibits reddish to black outcrops from clinker and burned sandstone due to natural burning of coal.
	Coniacian			Unconformity
	Turonian	Straight Cliffs Formation, Lower, Smoky Hollow Member *Minor Coal*	24-500?	Interbedded white or yellow-gray sandstone, light gray to dark gray mudstone and sometimes thin coal seams; lower part ledge to slope forming, upper part cliff forming.
		Straight Cliffs Formation, Lower, Tibbet Canyon Member	70-185	Yellow-gray and gray-orange, medium-grained sandstone interbedded with subordinate gray mudstone; cliff former.
		Tropic Shale *Minor Coal*	550-1,000	Medium- to dark-gray argillaceous to sandy shale, contains thin yellow-gray sandstone beds at top and base, otherwise uniform; forms badlands and slopes.
	Cenomanian	Dakota Formation *Minor Coal*	0-250	Yellow-gray sandstone alternating with gray shale, carbonaceous shale and coal; forms semiresistant ledge.

FIGURE 2-1: Stratigraphic Section of Cretaceous Strata Exposed in the Kaiparowits Region (Doelling and Graham, 1972)

SYSTEM	SERIES	STRATIGRAPHIC UNIT	THICKNESS (Feet)	DESCRIPTION
CRETACEOUS	Cenomanian	Dakota Formation Minor Coal	0-250	Yellow-gray sandstone alternating with gray shale, carbonaceous shale and coal; forms semiresistant ledge.
		Tropic Shale Minor Coal	250-1,800	Medium- to dark-gray argillaceous to sandy shale, contains thin yellow-gray sandstone beds at top and base, otherwise uniform; forms badlands and slopes.
	Turonian	Tibbet Canyon Member	18-185	Yellow-gray and gray-orange, medium-grained sandstone interbedded with subordinate gray mudstone; cliff former.
		Smoky Hollow Member Minor Coal	26-5007	Interbedded white or yellow-gray sandstone, light gray to dark gray mudstone and sometimes thin coal seams; lower part ledge to slope forming, upper part cliff forming.
		Allen Valley Member Major Coal Seams	500-900	Interbedded yellow-gray, white and orange medium-grained sandstone, gray shale, carbonaceous mudstone and coal; forms ledgy outcrops; often exhibits reddish to black outcrops from clinker and burned sandstone due to natural burning of coal.
		Drip Tank Member	180-354	Yellow-brown to gray-orange, fine- to medium-grained sandstone with some siltstone and conglomerate interbedded with subordinate gray shale; resistant cliff former.
		Naturita Formation	260-1,126	Yellow-gray resistant sandstone, siltstone and conglomerate alternating with yellow-orange nonresistant sandstone and gray mudstone; lower half dominantly nonresistant, upper half massive and hard.
		Kaiparowits Formation	2,000-5,500	Gray to dark gray, fine- to moderately coarse-grained, "salt and pepper" arkosic sandstone with subordinate light gray mudstone; weak calcareous cement, forms badlands and slopes.

FIGURE 3-1: Stratigraphic Section of Cretaceous Strata Exposed in the Kaiparowits Region
(Doelling and Graham, 1972)

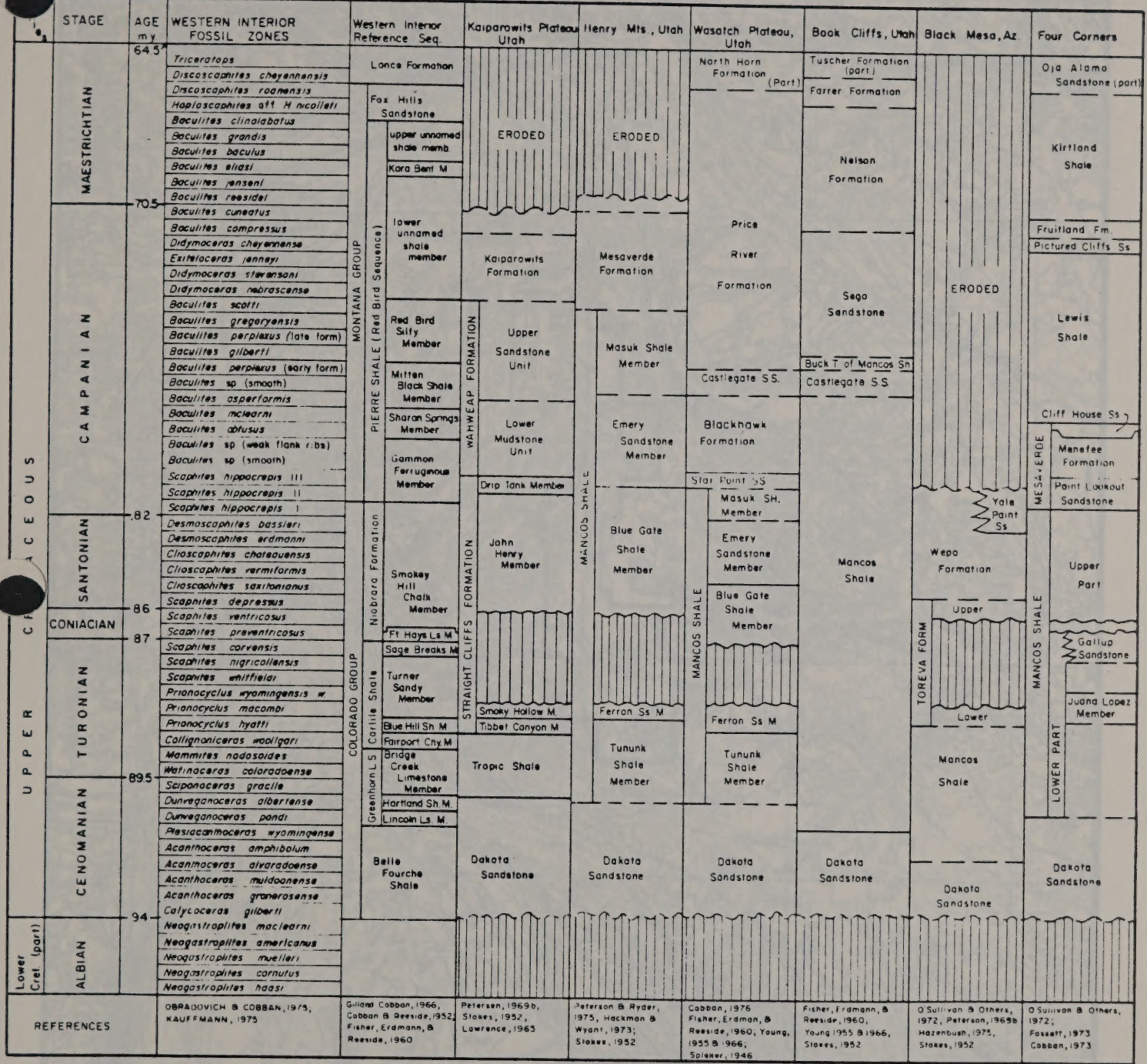
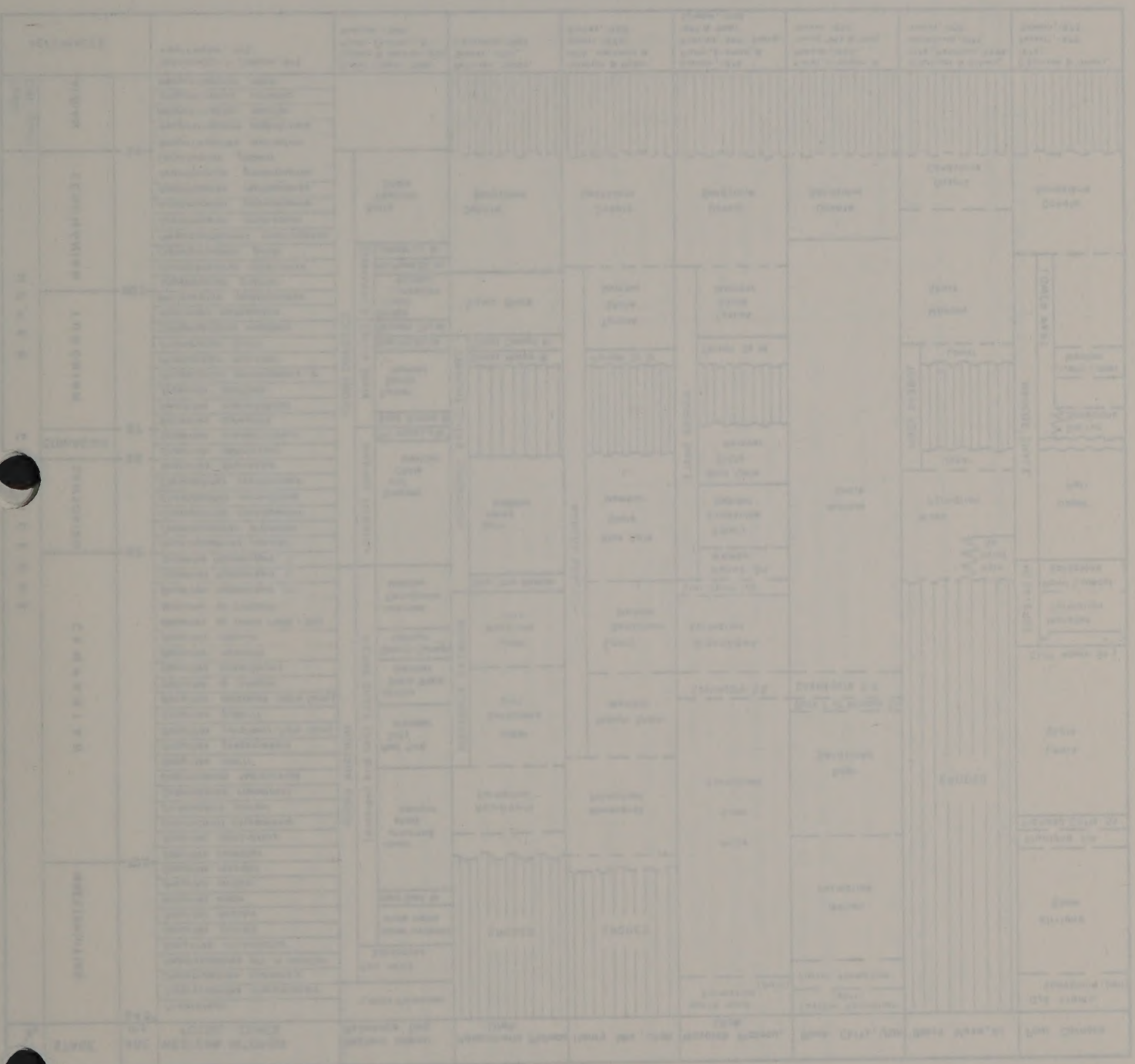
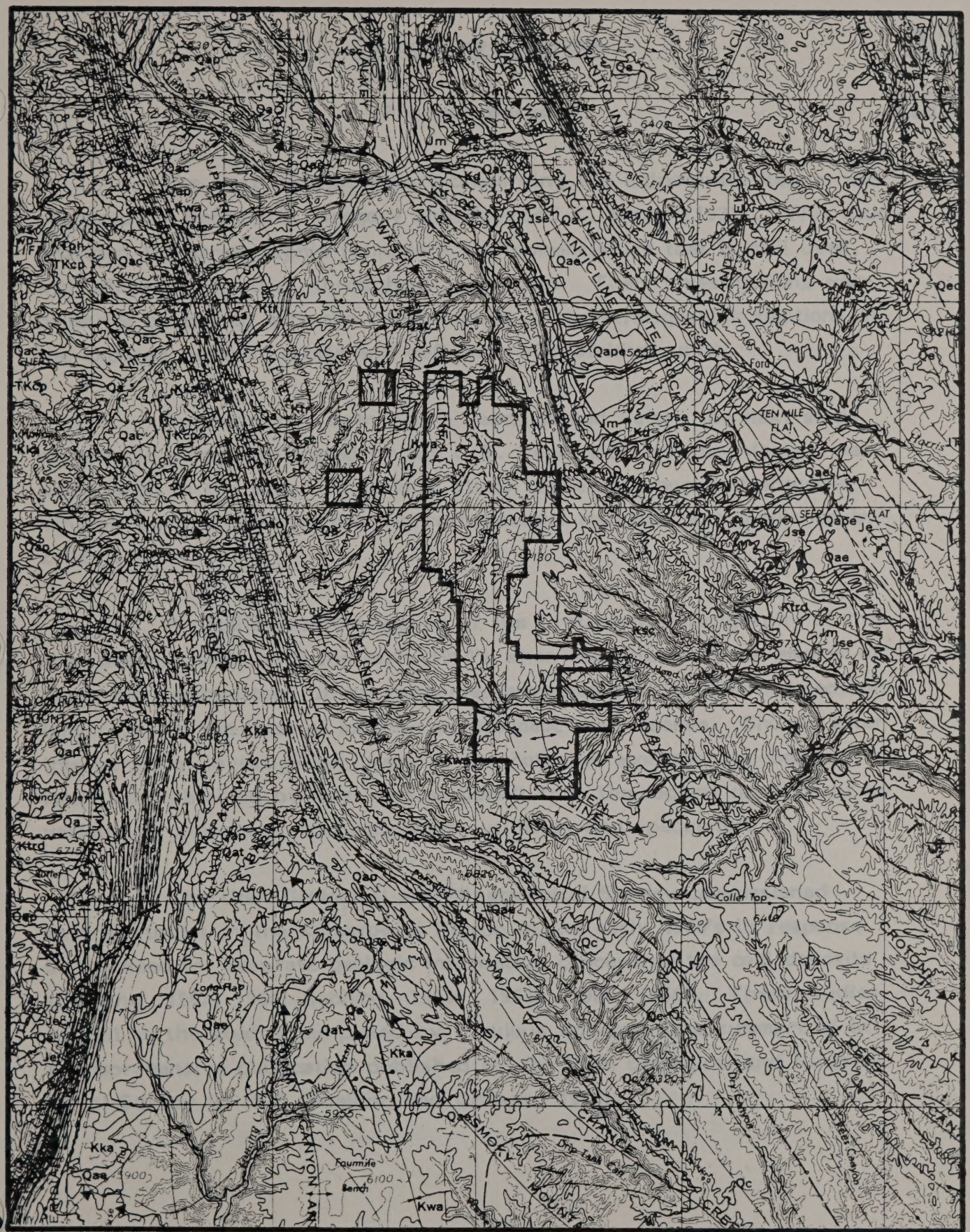


FIGURE 2-2: Regional Correlation Diagram for the Cretaceous Sediments of the Garfield Deposit

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Previous Workers 2.21

The first geologic report concerning the Kaiparowits Region was prepared by Herbert E. Gregory and Raymond C. Moore (1931). This study provided the stratigraphic framework and nomenclature that is still used, with minor modification, throughout the region. Their investigations were primarily concerned with the determination of the stratigraphic relations of the exposed Mesozoic strata and definition of individual units.

Approximately twenty-five years passed before the next stage of geologic study, instigated by the U.S. Department of the Interior, was undertaken in the area. A geologic study concerning non-coal economic minerals such as uranium-bearing carbonaceous materials was published by Zeller (1955).

A detailed reinvestigation of the Cretaceous stratigraphy of the area was not made until the early 1960's when individuals in the employ of numerous southwestern utility companies examined the coal-bearing sedimentary rocks in the hope of locating an economic coal reserve. Shortly after this period of investigation, many of the potential economic coal-bearing lands were obtained by lease and/or permit from the federal government by various oil companies, coal companies, utilities, and speculators.

The Utah Geological and Mineralogical Survey commissioned several studies in the Kaiparowits Region in the early 1960's in response to the need for accurate geologic information concerning the coal deposits (Doelling, 1967, 1968; Robison, 1963a, 1963b, 1964, 1966). These studies as well as the studies completed by previous workers were summarized in a comprehensive publication published by the Utah Geological and Mineralogical Survey in 1972 (Doelling and Graham, 1972).

The Department of the Interior also perceived the need to assess the value of the coal lands of the region which resulted in the commencement of mapping activities by the U.S. Geological Survey and Bureau of Mines in the early 1960's. The Bureau of Mines conducted a reconaissance geologic mapping program to determine the general geologic characteristics and coal reserves of the region (Grose, 1965; Grose, Hileman, and Ward, 1967) while the U.S. Geological Survey began quadrangle mapping in the Kaiparowits Region (Bowers, 1973; Pererson, 1966, 1973; Peterson and Barnum, 1973a, 1973b; Peterson and Horton, 1966; Peterson and Waldrop, 1966; Stephens, 1973; Waldrop and Peterson, 1966; Waldrop and Sutton, 1966a, 1966b, 1966c; Zeller, 1973a, 1973b, 1973c, 1973d; Zeller and Stephens, 1973). Much of the data collated by the U.S. Geological Survey workers has been summarized in more general form (Peterson, 1969a, 1975; Peterson and Waldrop, 1965). Peterson's doctoral thesis at Stanford University (Peterson, 1969b) emphasizes the geologic framework and depositional environments of the southern Kaiparowits Region and summarizes much of the information obtained by the U.S. Geological Survey.

Concurrent with the investigations of the outcropping coal-bearing strata were extensive drill coring programs undertaken by companies holding subsurface rights in the area. Among these companies were Resources Company, Atlantic-Richfield, Consolidation Coal Company, Peabody Coal Company, Delcoal Company, El Paso Natural Gas Company, Utah Power & Light Company, Sun Oil Company, and Woods Petroleum Company. Significant quantities of subsurface data concerning the relations of the coal-bearing strata were obtained during these programs.

Other studies of a more general nature concerning the oil potential, subsurface geology, and structure of the region were published in the late 1950's to middle 1960's (Bissell, 1954; Heylmun, 1958; Kunkel, 1965; McFall, 1971a, 1971b; Peterson and Graham, 1971; Stokes and Heylmun, 1965; Van de Graff, 1963). A map of the Escalante

Quadrangle (1:250,000) showing the distribution of geologic strata, structure, and uranium deposits for the area was published recently (Hackman and Wyant, 1973). Despositional analyses of the region were summarized by Vaninetti (1978,1979).

Drilling Programs

In addition to the mapping activities conducted by the workers noted above, a significant body of geologic information has been obtained by the numerous companies that have conducted drilling programs in the region since the early 1960's. The most extensively drilled portions of the Kaiparowits Plateau are located to the south of UP&L's deposit on leases controlled by a consortium of utilities and by El Paso Natural Gas Company.

Drilling activities have not been as extensive on the Garfield Deposit as for the previously mentioned cases. The Delcoal group, the original owners of a majority of the federal lands in the Garfield Deposit, completed eight drill holes in 1970 (Appendix 2-1). The data from these holes are nearly useless because of the lack of cores, descriptions, or geophysical logs and the fact that most holes were bottomed after penetrating only one of the uppermost coal seams. The 24 drill holes completed by UP&L were all cored and described in detail. Geophysical logs were obtained for 13 of the holes. Three of the holes were bottomed prematurely because of drilling problems. The data from these holes provide the bulk of the information available concerning the detailed correlation of coal zones, coal seams, and reserves within the Garfield Deposit.

Methods

As in most geologic evaluations, both field and office compilation activities were utilized to gain a full understanding of the geology of the deposit. Field methods consisted of both drilling programs and outcrop mapping activities. Office compilations consisted of collating field data and the preparation of maps, tabulations, and more interpretive illustrations.

As noted above, the significant drill hole data were obtained for 24 drill holes in 1971 and 1972. The locations for all drill holes are shown on the map in Appendix 2-1. The determination of drill hole bottom depths and core points were based on the outcrop information compiled by H.D. Zeller of the U.S.G.S. from the four quadrangles in which the deposit is located that had been open-filed just prior to the drilling program. These quadrangles, Carcass Canyon, Death Ridge, Canaan Creek, and Dave Canyon, have since been released as final maps (see Zeller, 1972a, 1972b, 1972c, 1972d).

Drill hole cores were obtained for all drill holes in a zone from 400' to 600' thick in the coal-bearing portion of the Straight Cliffs Formation. Thirteen of the holes were geophysically probed with natural gamma, density, and resistance geophysical equipment at a scale of 1" = 5' (copies of all geophysical logs have been submitted previously). Drill cores were collected, washed, and placed in core boxes by UP&L geologists to insure proper orientation of cores, determination of recovery factor, and proper bagging of coal samples. For the most part, all drill holes were geophysically probed by geologists.

Cores were transported to our office facility in Escalante where they were described to the nearest one-tenth of a foot. Five major rock types were recognized and described: sandstone, interbeds*, mudstone, carbonaceous mudstone, and coal. Copies of the lithologic descriptions for all drill holes have been submitted previously. When available, cores were

*Interbeds is used in this report as a separate and distinct lithologic term that represents sequences of interbedded sandstone, siltstone, and mudstone.

described in conjunction with the geophysical log for the hole described.

Once cores were described, they were photographed three times each and stored in slide carousels (which are available for inspection at UP&L's offices). Shortly thereafter (1 hour), significant coal cores were sampled, crushed, and split. A total of 161 samples were processed in this manner. All samples were analyzed for proximate tests, ash fusion temperatures, specific gravities, and Hardgrove grindabilities by Commercial Testing and Engineering of Denver, Colorado. Great care was taken to insure that all the moisture in the coal cores, including drilling fluids, were retained with the cores for accurate moisture determinations. Splits of these samples were analyzed in 1978 for the mineral analysis of ash by Standard Laboratories, Inc.

Lithologic descriptions and geophysical logs were later reduced to graphic form with the preparation of columnar sections for each of the drill holes. Each of these sections was drafted at a scale of 1" = 10' and a standard group of symbols were used to indicate rock types and significant features (Figure 2-4). Columnar sections for each of the 24 core holes as well as the eight holes drilled by the Delcoal group are included in Appendix 2-3.

At a much later date, UP&L began the collation of the drill hole data for the Garfield Deposit. Columnar sections were reduced to a scale of 1" = 20' and numerous north-south and east-west cross sections were constructed (Appendix 2-2). The first phase of cross-sectioning permitted the identification and recognition of the three major coal zones in the subsurface that had been recognized by Zeller at the outcrop. Later phases of cross-sectioning dealt with the correlation of individual coal seams and "blanket" sandstone beds. Two individuals working simultaneously, but separately, prepared cross-sections and developed correlations to serve as a check on the validity of a given correlation and to minimize biases. For the most part, the correlations developed by each individual were identical.

UTAH POWER & LIGHT COMPANY

ESCALANTE PROJECT

KEY TO COLUMNAR SECTIONS

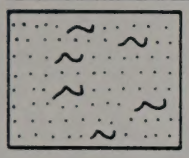
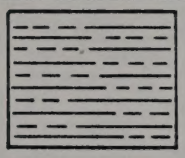
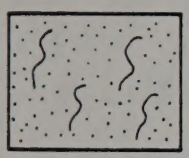
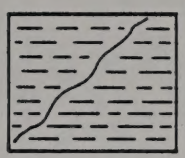
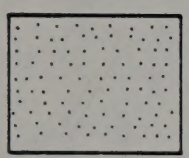
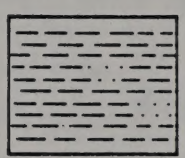
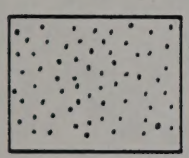
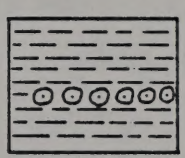
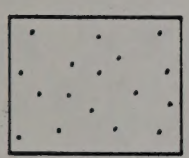
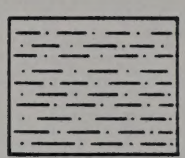
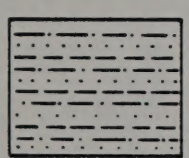
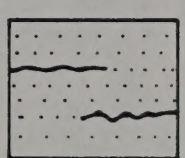
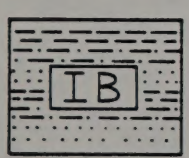
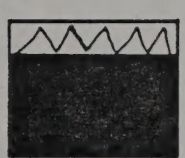
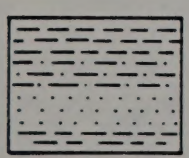
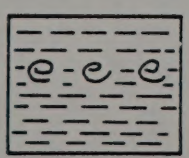
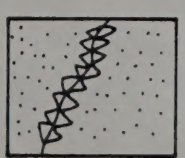
PATTERNS	LITHOLOGY	PATTERNS	LITHOLOGY
	TURBID SANDSTONE		CARB. COALY MUDSTONE
	SOFT, LOOSE SAND		COAL DIKE IN MUDSTONE
	LAYERED SANDSTONE (GASEOUS)		SANDSTONE INCLUSION IN MUDSTONE
	MASSIVE SANDSTONE (MEDIUM GRAINED)		CONCRETIONS
	MASSIVE SANDSTONE (COARSE GRAINED)		SILTSTONE
	INTERBEDS (FINE)		COAL VEINLETS IN SANDSTONE
	INTERBEDS (MEDIUM)		BONEY COAL
	INTERBEDS (COARSE)		COAL
	FOSSIL ZONE		FRACTURE FILLING OF MARCASITE

FIGURE 2-4: Key to Lithologic Symbols Used on Columnar Sections

UP&L reconnaissance mapping in the northern part of the Kaiparowits Plateau with a heavy emphasis on the outcrops exposed in Right and Left Hand Collet Canyons and in Alvey Wash and field trips with Tom Ryer (U.S.G.S.), John Horne (University of South Carolina) and the U.S.G.S. workers most familiar with the area (Zeller and Peterson) indicate that the environments in which the sediments were deposited can be recognized and traced in the outcrops, throughout the area. This realization indicates that the same depositional environments can probably be traced into the subsurface and recognized in the drill hole cores. Subsequent outcrop and compilation activities have therefore been oriented toward developing an understanding of the genesis of the deposit so that correlations and other interpretations are reasonable in terms of the depositional history of the deposit.

The emphasis placed on the recognition of depositional sequences and environments resulted in the visual examination of core slides in order to recognize sediments deposited in specific environments. Environmental designations were then transferred to the columnar sections and the cross-sections. Additional field mapping activities and examinations of cores were geared to recognizing typical depositional "packages" and evaluating the lateral continuity of each "package".

Cross-sections were modified in order to reflect the new insights gained as a result of a knowledge of the depositional processes involved in the accumulation of the deposit. The most promounced changes in cross-sections occurred in the southern one-third of the deposit. The cross-sections were then used to develop fence diagrams.

Fence diagrams were originally constructed at a vertical scale of 1" = 100' in order to illustrate all three coal zones within the Straight Cliffs Formation (Appendix 2-4). Later refinements indicated the necessity to construct fence diagrams for each of the coal zones at a vertical scale of 1" = 20' in order to minimize confusion in examining the correlations (see Appendix 2-5).

Once all fences had been completed, seam designations were developed using a numerical/alphabetical hierarchical code. The seam nomenclature will be discussed in a later section of the text.

Tabulations summarizing the pertinent data concerning the seam designation, seam thickness, depth in drill hole, interval between seams, and elevation above seal level were prepared for each drill hole (Appendix 2-6). Data from these tabulations, referred to as "Coal Summaries", were used to prepare isopach maps for each of the major coal seams as well as the intervals between the seams. Copies of these maps are included in Appendices 2-7 and 2-8, respectively. Data from the Coal Summaries were also used to prepare two structure contour maps (Appendix 2-9) and two overburden maps for the deepest and shallowest coal seams (Appendix 2-10).

Geologic Analysis 2.3

Structure 2.31

The structure in the area of the Garfield Deposit is characterized by several, widely spaced, north-south oriented anticlines and synclines. For the most part, dips on these structures rarely exceed 10° and usually average from 2° to 5°. All of these structures are characterized by broad anticlinal crests and synclinal troughs.

The Alvey Syncline, whose axis roughly parallels the western boundary of the Garfield Deposit, is the most dominant structure present in the area of UP&L's coal lands (Plate 2-1). The axis of the syncline runs the length of the deposit and plunges to the south. Dips into the syncline are generally less than 5° although a reading of 10° has been obtained in the northeastern corner of the deposit along a dip-slope as the Alvey Syncline transitionally passes into the Escalante Anticline.

A minor anticlinal structure, known as the Relishen Anticline, is located in the southeastern portion of the deposit. Dips generally range up to 5° on the flanks of the flexure within the confines of the deposit. The axis of the Relishen Anticline trends roughly north-south and is oriented convex to the west.

Characteristic of the Kaiparowits Plateau, there is an absolute lack of faulting on the Garfield Deposit. The only fault offset that has yet been detected is a minor fault located about three miles west of the deposit in the area of Canaan Creek/Coal Bed Canyon.

Stratigraphy 2.32

The general stratigraphic framework of the Kaiparowits Region

2.3 Structure

The structure in the area of the Garfield Deposit is characterized by several, widely spaced, north-south oriented anticlines and synclines. For the most part, dips on these structures rarely exceed 10° and usually average from 2° to 5°. All of these structures are characterized by broad anticlinal crests and synclinal troughs.

The Alkey Syncline, whose axis roughly parallels the western boundary of the Garfield Deposit, is the most dominant structure present in the area of UPTL's coal lands (Plate 2-1). The axis of the syncline runs the length of the deposit and plunges to the south. Dips into the syncline are generally less than 5° although a reading of 10° has been obtained in the northeastern corner of the deposit along a dip-slope as the Alkey Syncline transitionally passes into the Escalante Anticline.

A minor anticlinal structure, known as the Rollason Anticline, is located in the southeastern portion of the deposit. Dips generally range up to 5° on the flanks of the fissure within the confines of the deposit. The axis of the Rollason Anticline trends roughly north-south and is arcuate convex to the west.

Characteristic of the Kaiparowits Plateau, there is an absolute lack of faulting on the Garfield Deposit. The only fault offset that has been detected is a minor fault located about three miles west of the deposit in the area of Canaan Creek/Coal Bed Canyon.

2.32 Stratigraphy

The general stratigraphic framework of the Kaiparowits Region

PLATE 2-1:

Aerial view to west across the Straight Cliffs Escarpment into the Little Valley portion of the Garfield Deposit.

Note the laterally continuous littoral sandstone beds exposed at the outcrop. The stratigraphic units are noted on the overlay. Units above the Christensen Zone outcrop in the dip slope away from the viewer. The Alvey Syncline dominates the view area structurally.

CHRISTENSEN COAL ZONE

"B" SANDSTONE

"A" SANDSTONE

JOHN HENRY MEMBER

LOWER MARINE MUDSTONE

SMOKEY HOLLOW MEMBER

TIBBIT CANYON

TROPIC SHALE

has been discussed in a previous section and indicated that the major coal reserves of the Garfield Deposit are located within the Straight Cliffs Formation. Detailed mapping activities conducted by Peterson and other U.S.G.S. workers (notably Zeller, Waldrop, and Bowers) have resulted in the definition of four members within the Straight Cliffs Formation which are named, in ascending stratigraphic order, the Tibbet Canyon, Smokey Hollow, John Henry, and Drip Tank Members (Peterson 1969a). The age, thickness, and lithologic characteristics of each of these members are illustrated in Figure 2-1 and are discussed in the following sections.

Tibbet Canyon Member

The basal member of the Straight Cliffs Formation is composed nearly entirely of sandstone and weathers to form precipitous vertical cliffs above the Tropic Shale (Plate 2-1). The member ranges in thickness from 100 to 150 feet in the area of the Garfield Deposit, and consists of fine- to very fine-grained sandstone and mudstone in its lower portion which grades transitionally into medium-grained sandstone in its upper portion. The Tibbet Canyon Member interfingers with and conformably overlies the Tropic Shale and is conformably overlain by the Smokey Hollow Member. Depositional studies by Peterson (1969b) and Vaninetti (1979) indicate that the Tibbet Canyon Member accumulated in littoral marine environments along the front of an advancing deltaic complex during the first major regression in the Upper Cretaceous Seaway of the Western Interior. Similar interpretations have been forwarded by workers who have investigated stratigraphic and age equivalent units such as the Ferron Sandstone, Frontier Formation, and Gallup Sandstone in adjoining states.

Smokey Hollow Member

The Smokey Hollow Member ranges in thickness from 100 to 270 feet and consists of sandstone, interbeds, mudstone and coal. The member is unconformably overlain by the John Henry Member and weathers to form a slope capped by a cliff-forming sandstone (Plate 2-1).

The Smokey Hollow Member was deposited in subaerial continental environments located behind the shoreline of the "Tibbet Canyon Delta". The environments which can be recognized, in ascending stratigraphic order, are lagoonal-paludal, swamp, meander-belt fluvial flood plain, and braided-belt fluvial (Peterson 1969b, Vaninetti 1979), thin coal seams (less than 4' thick) are present in the Smokey Hollow. The uppermost unit of the Smokey Hollow is a braided belt sandstone, referred to informally as the "calico bed", which marks the unconformity between the Smokey Hollow and the John Henry Member. This "calico bed" is bleached white, due to feldspar alteration, and is therefore easily recognizable on the outcrop.

John Henry Member

The John Henry Member is composed of sandstone, mudstone, interbeds, and the major coal seams within the Kaiparowits Region and weathers to form a steep slope (Plate 2-1). It ranges in thickness from 880 to 1035 feet thick and averages about 950 feet thick on the Garfield Deposit. The member intertongues with and is conformably overlain by the Drip Tank Member of the Straight Cliffs Formation.

Mapping by Peterson (1969a, 1969b) indicates that numerous facies changes occur within the John Henry Member which complicates understanding the stratigraphic relations of the unit within the Kaiparowits Region. Peterson's analyses of the John Henry Member (Peterson 1969a, 1969b) indicates that seven laterally continuous barrier island sandstone bodies were deposited on top of one another in the eastern part of the Kaiparowits Plateau and that the major coal zones encountered in the central part of the plateau were deposited in swamps located landward of the barrier islands (Figure 2-5). The seven barrier island sandstone beds are informally referred to as the "A" through "G" sandstones and the three major coal zones that are located to the west (or landward of the sandstone bodies) are termed the Christensen, Rees, and Alvey Coal Zones.

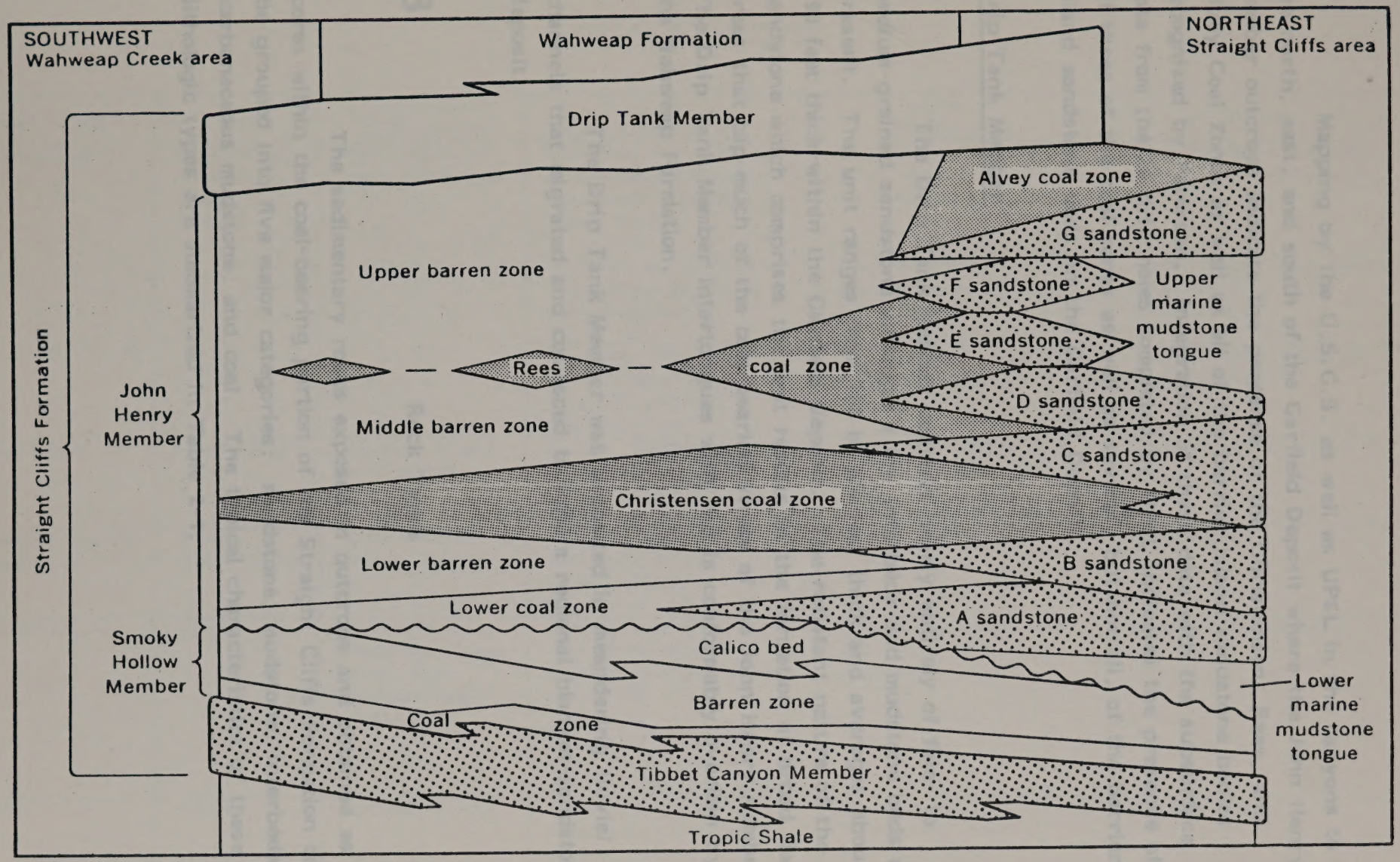


FIGURE 2-5: Generalized cross section through the Kaiparowits Plateau (Peterson, 1969a). The Garfield Deposit is generally located in the right-hand side of the diagram.

Mapping by the U.S.G.S. as well as UP&L in the canyons to the north, east, and south of the Garfield Deposit where the John Henry Member outcrops indicate the presence of the Christensen, Rees, and Alvey Coal Zones as well as all of the barrier island sandstone bodies recognized by Peterson. Integration of these data with the subsurface data from the 24 drill holes completed by UP&L indicates the presence of all three of the coal zones as well as several, but not all, of the barrier island sandstone beds on the Garfield Deposit.

Drip Tank Member

The Drip Tank Member consists nearly entirely of fine- to medium-grained sandstone although a few interbeds and mudstone beds are present. The unit ranges from 160 to 220 feet thick and averages about 190 feet thick within the Garfield Deposit. The resistant nature of the sandstone which comprises the unit results in the formation of broad flat areas that cap much of the coal-bearing strata of the John Henry Member. The Drip Tank Member intertongues with and is conformably overlain by the Wahweap Formation.

The Drip Tank Member was deposited in meandering fluvial channels that migrated and coalesced to form a regional blanket-sandstone deposit.

2.33 Rock Types

The sedimentary rocks exposed in outcrops and obtained as cores within the coal-bearing portion of the Straight Cliffs Formation can be grouped into five major categories: sandstone, mudstone, interbeds, carbonaceous mudstone, and coal. The typical characteristics of these lithologic types are summarized in Table 2-1.

TABLE 2-1

TYPICAL ROCK TYPES

Rock Type	Color*	Structure-Competence	Typical Characteristics	Local Features	Contacts
Sandstone	Light-Gray Dark-Gray Gray Buff	Well cemented; very competent	Massive; very fine to medium-grained; "salt and pepper"; lense-shaped	Mudstone clasts at base; upward decrease in grain size	Base is sharp and erosional; top gradational
			or		
			slumped to mottled; very fined-grained; silty interbeds	Root mottling; iron-rich concentrations; bioturbation	Base ranges from sharp to gradation; top gradational
			or		
			massive; fine-to medium-grained; laterally continuous "blanket"	bioturbation; upwards increase in grain size	Top is sharp; base is gradational
Mudstone	Dark-Gray Gray Black-Gray	Fissile; slacks with water	Dense, homogeneous; structureless; local compactional slickensides	Tan, very dense and hard iron-rich nodules; silty interbeds, coaly veinlets	Gradational to sharp
Carbonaceous Mudstone	Black Black-Gray	Slacks with water, fissile	Dense, coaly veinlets; brittle to fissile, plant impressions	Resin, fossils	Gradational
Interbeds	Variable	Partings along bedding planes when well-bedded; well cemented	Mottled and slumped; thin to thickly bedded; sandy, silty, or muddy	Tan, very dense and hard iron-rich nodules	Sharp to gradational
			or		
			horizontally bedded; very thinly bedded; sandy, silty, or muddy		
Coal	Black	Brittle; cleats	Bright; vitreous; resinuous; marcasite and/or calcite on cleats and at top of base	Boney zones; "blistered" coal; 1/4" ash beds	Usually sharp

*Most common color at top

Sandstone

Sandstone is one of the more abundant rock types present within the Garfield Deposit particularly so at outcrop where it weathers to form resistant ledges and cliffs (Plate 2-1). It is typically fine- to medium-grained, light-brown to tan or white, and exhibits significant variations in sorting, rounding, matrix content, and sedimentary structures.

Genetically, the sandstone within the deposit can be classified as either fluvial or littoral. Littoral sandstone beds can be further subdivided into lower shoreface, upper shoreface, and foreshore units. The characteristics of each of these types of sandstone are summarized in Table 2-2.

Mudstone

Mudstone is readily recognizable in core but is rarely exposed in outcrop since it tends to form slopes which are usually covered with talus. Mudstone is usually dark gray, dense, homogeneous and structureless. Plant impressions are common in continental mudstone associated with fluvial sandstone while offshore marine mudstone is structureless and is extensively bioturbated. Cores commonly exhibit compactional slickensides in mudstone.

Carbonaceous Mudstone

Carbonaceous mudstone is best exposed in core and occasionally rare outcrop exposures. It is typically black to gray-black, fissle, dense, homogeneous, structureless, and contains compactional slickensides. Carbonized plant impressions and significant amount of carbonaceous material are common. Carbonaceous mudstones are only associated with coal seams.

TABLE 2-2

CHARACTERISTICS OF SANDSTONES

Sandstone Type	
Fluvial	Upward fining; medium-high angle trough cross-stratification grading into ripple laminated; angular-sub rounded; poorly sorted; clay clasts along bedding; erosional basal contact, gradational upper contact; lense shaped; typically 15' thick; irregular bedding; rare bioturbation; weathers to ledges.
Foreshore	Upward coarsening, gradational lower contact sharp upper contact; very regular bedding; horizontal laminnae; very well sorted, winnowed, well rounded, typically bleached white; tabular bedding, 10-20; thick; weathers to ledges, herringbone stratification.
Upper Shoreface	Upward coarsening; gradational upper and lower contacts; low angle, med-small scale trough cross-stratification; rare burrows; fair sorting; rounded grains; tabular; 20-30' thick; weathers to cliffs; very regular bedding.
Lower Shoreface	Upward coarsening; regular bedding; convolute hummocky and low angle med-small scale trough stratification; burrowed, sharp to gradational basal contact; rounded; moderate sorting; tabular, 15-40' thick; weathers to cliffs.

Interbeds

Interbeds are best exposed in core and rarely in outcrop. They consist of thinly bedded units of sandstone, mudstone and siltstone. They are usually structureless, mottled and compacted. Many contain dense, hard, indistinct iron-rich nodules. They are deposited in both offshore and marsh environments.

Coal

Coal, the most significant rock type in the Garfield Deposit is characterized by a black, lusterous appearance, brittleness, vertical fractures, low specific gravity (1.35 to 1.45), resinous and dull bands, and marcasitic and/or gypsiferous fracture fillings. Coal tends to have impurities such as bone, rare ash beds and carbonaceous mudstone interbedded with them.

Sedimentation

Depositional Model

The sedimentary rocks which comprise the coal-bearing John Henry Member of the Straight Cliffs Formation were deposited in a strandplain/barrier island setting that trended about N 20° W (Zeller, 1972d) on the western margin of the Upper Cretaceous Seaway of the Western Interior as documented by Peterson (1969b), McGookey (1972), and Vaninetti (1978, 1979). A depositional setting consisting of several parallel belts that parallel the shoreline wherein specific lithologic associations are found characterizes the sediments of the John Henry Member (Figure 2-6). The depositional belts that have been reconstructed for the sediments of the Kaiparowits Plateau consist of offshore, transition, lower shoreface, upper shoreface, foreshore, lagoon, swamp, marsh, and fluvial-floodplain belts. The characteristic features of each of these depositional belts are summarized in Figure 2-7.

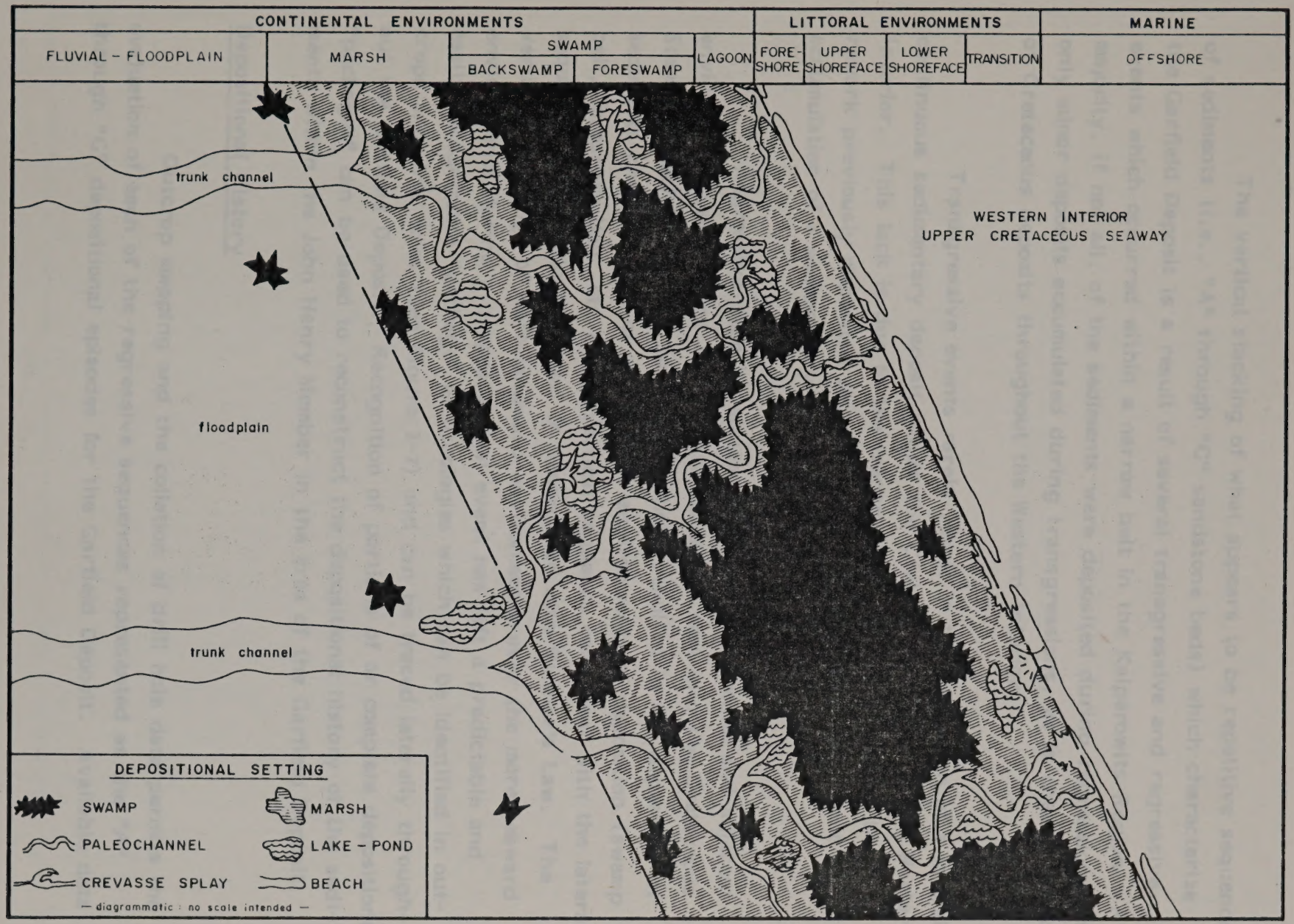


Figure 2-6: Depositional Model for the Kaiparowits Plateau

The vertical stacking of what appears to be repetitive sequences of sediments (i.e., "A" through "G" sandstone beds) which characterize the Garfield Deposit is a result of several transgressive and regressive events which occurred within a narrow belt in the Kaiparowits Region. A majority, if not all, of the sediments were deposited during regressions and only minor deposits accumulated during transgressions, as is characteristic of Cretaceous deposits throughout the Western Interior.

Transgressive events rarely result in the deposition of a laterally continuous sedimentary deposit within the Cretaceous section of the Western Interior. This lack is due to the transgressive process whereby the seas rework previously deposited sediment at a rate that does not permit the accumulation of a sedimentary deposit.

Regressive deposits accumulated as the shoreline and adjacent environments (see Figure 2-6) prograded to the east into the Cretaceous Seaway. Each regression (or progradation) resulted in the deposition of sediments characteristic of each of the environments of deposition (swamp belt, foreshore belt, etc.) in a vertical sequence consistent with the lateral sequence of depositional "belts" in accordance with Walther's Law. The vertical "stacking" of more landward environments over the more seaward environments during a progradational event has left a predictable and easily recognizable "package" of lithologies which can be identified in outcrops and drill holes (see Figure 2-7) and can be traced laterally throughout the Garfield Deposit. Recognition of portions of or complete depositional "packages" can be used to reconstruct the depositional history of the sediments within the John Henry Member in the area of the Garfield Deposit.

Depositional History

Outcrop mapping and the collation of drill hole data permits the evaluation of each of the regressive sequences represented as the "A" through "G" depositional episodes for the Garfield Deposit. Available data

FACIES	DESCRIPTION	GRAPHIC LOG	FACIES CONTAINED WITHIN EACH REGRESSION ON THE GARFIELD DEPOSIT
FLUVIAL-FLOODPLAIN	Sandstone; Fines Upward; Erosional Base; Poorly Sorted; Trough And Micro-Cross - Bedded; Scour And Fill Structures; Lense-Shaped. Mudstone And Interbeds; Light Gray; Slope Former		
MARSH	Mudstone And Interbeds; Local Carbonaceous Material; Local Fluvial Channels; Slope Former		
SWAMP	Coal Seams, Carbonaceous Mudstone, Local Fluvial Channels And Interbeds; Slope Former; Burn Common		
LAGOON	Carbonaceous Mudstone, Oyster Reefs; Thinly Bedded; Local Sandy Washover Fans; Slope		
FORESHORE	Sandstone; Medium-Grained; Massive Cliff; Horizontal And Herringbone Bedding; White-Cap; Clean And Well Sorted; Regular Bedding		
UPPER SHOREFACE	Sandstone; Fine To Medium-Grained; Coarsens Upward; Trough Cross-Stratification; Very Regular Bedding; Low Angle Bedding; Minor Bioturbation		
LOWER SHOREFACE	Sandstone; Fine-Grained; Coarsens Upward; Hummocky And Trough Cross-Stratification; Bioturbated; Regular Bedding		
TRANSITION	Sandstone And Mudstone Beds; Very Fine-Grained Sandstone Beds With Sharp Basal Contacts; Profusely Bioturbated; Slope		
OFFSHORE	Mudstone; Homogeneous; Slope Former; Intensely Bioturbated		

FIGURE 2-7: Idealized Progradational Sequence Illustrating the Facies Contained Within Each Regressive "Package" for the Garfield Deposit.

indicate that the "B", "D", and "G" progradations were the only depositional episodes that permitted the development of a complete depositional "package" through the coal swamp environment (Figure 2-7). Other progradations were not as complete and as a consequence, well-developed coal seams did not accumulate. The fence diagram (Appendix 2-4) of the coal zones and major littoral sandstones illustrates the development of the marine and continental facies associated with the "B". "D", and "G" sandstone beds. Plate 2-2 illustrates each of the major stratigraphic units including the littoral sandstone beds and coal zones in the area to the east of the deposit. The depositional history for each of the seven major regressions in the Garfield Deposit are summarized below.

"A" Sandstone Regression: The "A" Sandstone Regression represents a fairly complete depositional sequence from the offshore to basal part of the coal swamp environment. This progradational package extends further landward (or to the west) than any of the other regressions (Figure 2-5). The poorly developed coal swamp facies and lack of marsh and fluvial-floodplain facies at the Garfield Deposit may indicate that the progradation may have reached its maximum seaward extent near the deposit and that the landward environments never prograded through the area. The 9.1 foot thick coal seam in one of UP&L's western-most drill holes (EP-DH-26) and the presence of well-developed coal seams in the drill holes completed by the company that controls the land adjacent to and west of UP&L's lands may indicate that a complete depositional cycle is present to the west of the Garfield Deposit.

"B" Sandstone Regression: The "B" Sandstone Regression is a well-developed depositional package that includes environments ranging from the transition zone to the swamp belt and contains the Christensen Coal Zone. The unit includes all but the offshore marine mudstone portion of the lower part of a cycle which probably indicates that the transgression that took place between the "A" and "B" sequences did not extend far enough landward to result in the deposition of the sediments of this environment when the "B" Regression began.

"D" Sandstone Regression: The "D" Sandstone Regression appears to be another well-developed progradational package of sediments that were deposited in environments that range from lower shoreface to the middle of the coal swamp. The Rees Coal Zone, which consists of six coal seams at the Garfield Deposit, accumulated during the "D" Sandstone Regression.

The lack of offshore and transitional sediments at the base of the sequence indicates that the preceeding transgression did not extend too far landward but far enough landward to result in the deposition of the "D" Sandstone across much of the northern two-thirds of the Garfield Deposit. The lack of the marsh and fluvial-floodplain environments in the area indicates that the "D" Sandstone did not prograde so far east into the Upper Cretaceous Seaway so as to allow the entrance of these more landward environments into the deposit.

"E" and "F" Sandstone Regressions: The "E" and "F" Sandstone Regressions are the two least understood progradational dequences in the area of the Garfield Deposit. Available data indicates that they are present only in the most seaward portions of the deposit in the vicinity of Right and Left Hand Collet Canyons and in the northern reaches of Alvey Wash. The lack of offshore, transition, and foreshore sediments and the presence of only the shoreface environments indicates that the transgressions that occurred between the "D" and "E" Regressions and the "E" and "F" Regressions did not extend into the Garfield Deposit to any significant degree. Further, both regressions appear to have been short-lived because of the lack of deposits from foreshore and more landward environments.

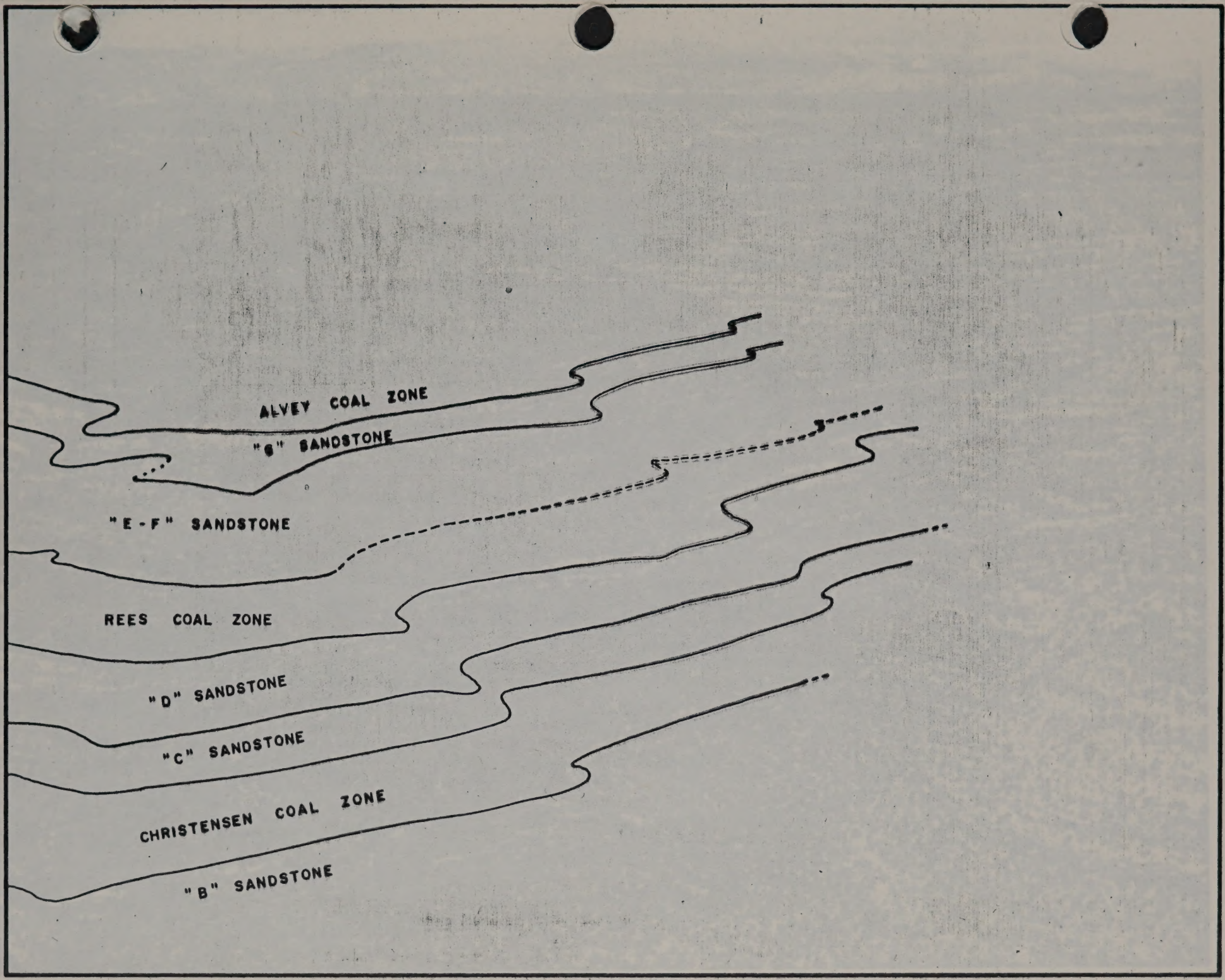
"G" Sandstone Regression: The "G" Sandstone Regression and Alvey Coal Zone represent a complete sequence of prograding environments that range from the transition zone to the fluvial-floodplain settings. The transgression that separates the "F" and "G" progradational episodes extended landward only to about the middle of the southern portion of the Garfield Deposit. The pinchout of the "G" Sandstone has been mapped in the tributary canyons of Right Hand Collet Canyon, the headwaters of Lower Trail Canyon (part of Left Hand Collet Canyon), and in the subsurface (Plate 2-3).

PLATE 2-2.

Exposure of the John Henry Member in Right Hand Collet Canyon.

Garfield Deposit located in middle background.

Note the lateral continuity of the littoral sandstone beds and coal zones.


ALVEY COAL ZONE
"G" SANDSTONE
"E - F" SANDSTONE
REES COAL ZONE
"D" SANDSTONE
"C" SANDSTONE
CHRISTENSEN COAL ZONE
"B" SANDSTONE

-B- SYNCLINE
CHANNELSBURG COVE ZONE
-C- SYNCLINE
-D- SYNCLINE
DEER COVE ZONE
-E-1-2- SYNCLINE
-F- SYNCLINE
WYMA COVE ZONE

PLATE 2-3.

Exposure of the upper part of the Straight Cliffs Formation in Right Hand Collet Canyon in the Garfield Deposit.

Sandstone beds of the Drip Tank Member cap this exposure. Alvey and Rees Coal Zones are exposed. Note the absence of the E, F, and G sandstone beds in this area.

The thin coal zone that is comprised of a single seam that locally is split into two subseams, the presence of numerous fluvial channels, and a significant quantity of non-carbonaceous mudstones and interbeds that overlies the "G" Sandstone may indicate that the progradation of the "G" Sandstone may have been too rapid to permit the accumulation of a well developed swamp. A stronger and more active fluvial influence on the deposition of sediments in this regression is suggested.

The progradation of the "G" Sandstone/Alvey Coal Zone resulted in the exit of marine and nearshore conditions from the Kaiparowits Region for the remainder of the Cretaceous. Fluvial and floodplain environments dominated the sedimentation in the region thereafter.

Depositional Mechanics

The mechanisms responsible for transgressions and regressions along the margins of the Upper Cretaceous Seaway of the Western Interior are thought to be related to tectonism, eustatic sea level changes, and depositional processes. Eustatic sea level changes and tectonic mechanisms affect the timing of regressions and/or transgressions on a widespread if not continental scale. Depositional processes affect sediment dispersal patterns on more of a regional scale.

Transgressive/regressive features in the Kaiparowits Region which occurred simultaneously at numerous other sites within the Western Interior are (1) the transgression of the Tropic Sea, (2) the regression of the Tibbet Canyon Member, and (3) the transgression that separates the Smokey Hollow from the John Henry Member. The initial marine transgression that resulted in the widespread development of the upper Cretaceous Seaway in the Western Interior occurred nearly simultaneously throughout the Western Interior and for that matter, can be recognized on a worldwide scale. The initial regression into the Tropic Sea (Mancos Sea) occurred at the same time all along the western boundary of the Upper Cretaceous Seaway and resulted in the

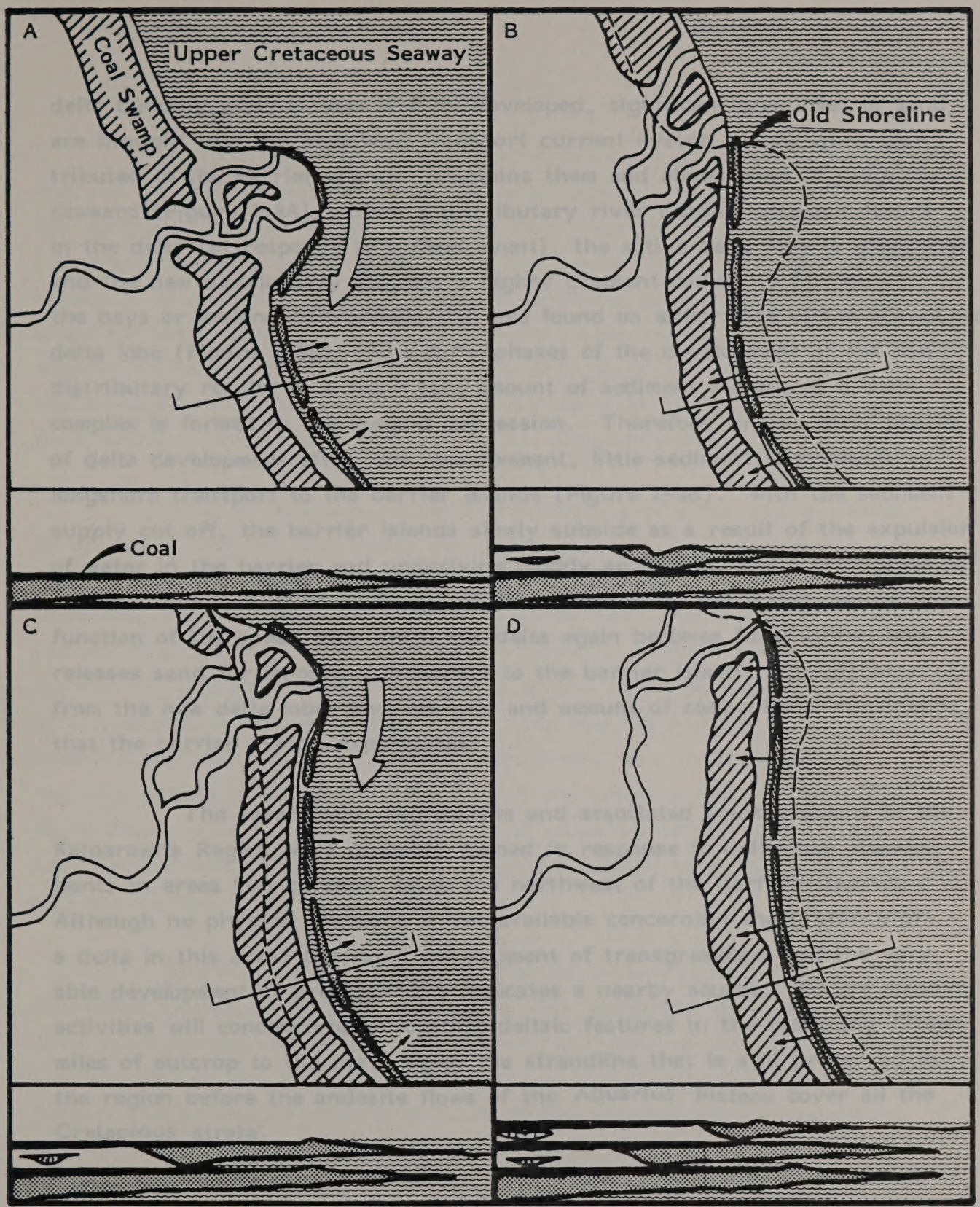


FIGURE 2-8: Depositional processes that result in transgressions and regressions in coal-bearing strata. Large arrow indicates time of significant longshore transport. Small arrows indicate direction of movement of shoreline. Cross-sections in strandplain area shown in lower part of panels.

delta building when a delta is fully developed, significant quantities of sand are brought into the longshore transport current system which, when distributed to the barrier islands, maintains them and allows them to prograde seaward (Figure 2-8A). When a distributary river channel avulsion occurs in the delta (in response to a flood event), the active delta lobe is abandoned and the new distributary assumes a higher gradient course to fill one of the bays or lowland depressions that are found on either side of the abandoned delta lobe (Figure 2-8B). The early phases of the development of the new distributary results in a significant amount of sediment storage as a delta complex is formed in the lowland depression. Therefore, in the early phases of delta development after lobe abandonment, little sediment is available for longshore transport to the barrier islands (Figure 2-8B). With the sediment supply cut off, the barrier islands slowly subside as a result of the expulsion of water in the barrier and underlying muddy sediments, and a transgression occurs. The amount of landward advance attained by the transgression is a function of the speed with which the delta again becomes fully formed and releases sand for longshore transport to the barrier islands, the distance from the new delta lobe, and the rate and amount of compactional subsidence that the barrier system experiences.

The seven major regressions and associated transgressions in the Kaiparowits Region were probably formed in response to delta lobe abandonments in areas "up current" or to the northwest of the Garfield Deposit. Although no physical evidence is yet available concerning the presence of a delta in this area, the rapid development of transgressions and the variable development of progradations indicates a nearby source. Future mapping activities will concentrate on locating deltaic features in the remaining fifteen miles of outcrop to the north along the strandline that is still preserved in the region before the andesite flows of the Aquarius Plateau cover all the Cretaceous strata.

The Garfield Deposit is characterized by a near lack of water that can be used to water stock or for other agricultural purposes. The few springs and seeps in the area have been developed by the grazing permittees for collection in stock ponds and tanks. The arid climate that results in an annual rainfall of less than 20 inches causes significant problems for the grazing permittees in providing water to their stock. A majority of the rainfall that occurs in the area takes place during summer thunderstorms when the dry washes, particularly Alvey Wash, experience flash floods. These washes remain dry for a majority of the year.

Water is sometimes found in the bottoms of a few of the washes in the area. For the most part, this water does not surface except when flash flood events occur. Water was obtained for drilling purposes by trenching the alluvium in the bottoms of Alvey Wash and Right Hand Collet Canyon (Camp Spring). A water sample collected in the wash at Camp Spring was from surface drainage in the canyon that was flowing at a rate of 8.6 gallons per minute (Table 2-4). This was the greatest water flow measurement encountered anywhere in the area of the Garfield Deposit which is probably indicative of the size of the drainage system that feeds Right Hand Collet Canyon. Local surficial concentrations of alkali water were encountered in the Left Fork of Carcass Canyon in June of 1979 but had since dried up by July of 1979.

The few seeps and springs that have been identified in the area are either located at contacts between massive sandstone beds and mudstone beds or in the bottoms of a few of the washes. Most of the springs on the Garfield Deposit are located at the contact of the thick mudstone unit at the base of the Wahweap Formation and the medium-grained massive sandstone beds at the top of the Drip Tank Member of the Straight Cliffs Formation. Samples were collected and flows measured for three springs at this contact in July of 1979 (Table 2-3) and for the most part flows were less than one

TABLE 2-3

WATER QUALITY

Name	Type	Flow	Temp °F	Oil & Grease	Fe	Date	Formation	Location
Oak Spring	Spring	.125 gal/min	66°	.1 mg/l	.01 mg/l	7/21/79	Ksd	NE¼ NE¼ Sec. 13, T. 36 S., R. 2 E.
Rock Spring	Spring	1 gal/min	59°	1 mg/l	.07 mg/l	7/21/79	Kw ss	NE¼ NE¼ Sec. 6, T. 37 S., R. 3 E.
Death Ride	Well	-	82°	.4 mg/l	.27 mg/l	7/23/79	Kw	Sec. 8, T. 37 S., R. 2 E.
Camp Spring	Spring Fed Stream	8.6 gal/min	82°	.1 mg/l	.21 mg/l	7/24/79	-	SW½ SW¼ Sec. 36, T. 37 S., R. 2 E.
Hard Head Water	Developed Spring	1 gal/min	61.5°	.1 mg/l	.1 mg/l	7/24/79	Kw	SE ¼ Sec. 15, T. 38 S., R. 3 E.

TABLE 2-4

TYPICAL SURFACE WATER ANALYSIS

SOUTHERN UTAH STATE WATER LABORATORY
SOUTHERN UTAH STATE COLLEGE
CEDAR CITY, UTAH 84720

RESULTS ON SAMPLE ANALYZED FOR:

Bureau of Land Management

COLLECTION DATE: March 31, 1977
SAMPLE NUMBER: 04150063
SITE LOCATION: Rock Spring, Garfield County, Escalante - T 37S, R 3E, Sec. 6

GENERAL PARAMETERS		CATIONS		CATIONS		ANIONS	
7176	pH	4.9	POTASSM	.013	LEAD	553.6	BICARB
1337	CONDUCT	15	SODIUM	<.01	MODBDNM	<.01	HYDROXID
1.5	TURBIDITY	94	CALCIUM	146.8	MAGNESM	320	SULFATE
839	HARDNESS	.185	AMMONIUM	.007	MANGANES	25.3	CHLORIDE
454	ALKALNTY	.013	ARSENIC	.008	NICKEL	.087	FLOURIDE
6.6	SUS SLD	.001	CADMIUM	.003	SILVER	.398	NITRATE
		.01	COPPER	.065	ZINC	.199	PHOS TOT
		.022	IRON TOT	<.01	MERCURY		

gallon per minute. A detailed analysis of the water at Rock Spring, which is at this same contact, was obtained from the Bureau of Land Management (Table 2-4).

Other springs occur at the base of the porous and permeable sandstones within the Straight Cliffs Formation. The Drip Tank Member and each of the "A" through "G" Sandstone beds probably have the porosity and permeability necessary to be aquifers if sufficient water was available to recharge them. One water sample was collected at the contact of the Drip Tank Member with the John Henry Member (Oak Spring) in July of 1979 (Table 2-3). The flow was barely measureable. Minor seeps noticeable from some of the littoral sandstone beds in June 1979 outcrop mapping had dried up in July of 1979.

Inasmuch as the Garfield Deposit is located in the axis of a syncline and the flanks of the syncline expose the coal-bearing strata to the surface, particularly so on the western flank, it is likely to assume that the porous and permeable littoral sandstones ("A" through "G") and the Drip Tank Member experience some degree of recharge. This water, coupled with the connate water that was trapped in these sandstone beds during deposition, has probably been concentrated in the synclinal axis of the Alvey Syncline in the laterally continuous sandstone beds that are interstratified with the coal-bearing strata.

The predominance of mudstone, interbeds, coal seams, and other fairly impermeable strata within each of the three coal zones has probably inhibited the influx of water into the coal zones. Meager hydrologic information from drill shaft reports indicates that this may be the case inasmuch as water was encountered in some of the littoral sandstone beds in drilling and lost circulation problems occured in several of the coal seams. These circulation problems indicate that the coal seams are dry but are fractured ("cleated") to the degree that water can pass through them. Circulation problems were so serious in drill holes 30 and 31 that the holes had to be abandoned prior to the complete penetration of the Christensen Coal Zone.

In summary, available data indicates that meager surface supplies of water (less than 10 GPM) are available in the bottoms of each of the major drainages in the region (Alvey Wash and Right Hand Collet Canyon) except during flash floods and that the laterally continuous sandstone beds within the Straight Cliffs Formation contain indeterminable amounts of connate and meteoric water from recharge areas to the west. The lack of significant seepage from these units indicates that only minor quantities of water are contained within them. The water within these sandstone units has not migrated into the coal zones to any appreciable degree because of the lack of permeability of the fine-grained sediments that make up the coal zone. Coal seams are probably dry or nearly so as judged by the ease with which water passes into them during drilling operations.

UP&L's experiences in mining the coals of central Utah in seemingly dry strata with only minor recharges indicate that the seepage that occurs as a result of fracturing of the sediments that enclose a mineable seam is sufficient to satisfy underground mining equipment requirements. This seepage originates from the porous but impermeable mudstone and perched sandstone beds that overlie coal seams. A similar experience is expected for the mining of the coal seams of the Garfield Deposit.

The only source of significant quantities of water in the area appears to be the Navajo Sandstone at a depth of about 4000 feet below the surface of Little Valley. Water from the Navajo Sandstone has been encountered in significant quantities in the drilling of the oil wells of the Upper Valley Oil Field.

Coal Seam Correlations 2.41

The stratigraphic relations of the coal-bearing strata within the Garfield Deposit were evaluated and significant units identified and correlated on the basis of stratigraphic position, the nature of enclosing sediments, geometry, similarities in adjacent drill holes, interpretation of depositional sequences and environments, and laterally continuous littoral sandstone units. Correlations were progressively developed from large-scale features such as the recognition of the laterally continuous sandstone beds and three coal zones to detailed correlations of individual seams within coal zones and rock splits within seams. The methods used to develop the correlation are more fully described in the methods section (2.23) of this report.

Correlations for the northern two-thirds of the deposit were fairly straightforward inasmuch as the littoral sandstones and coal zones were uniformly developed and easily understood. Coal seam correlations in that area are judged to have a reliability factor of about 90% or more. Correlations in the area to the south of drill hole #8 were more difficult to establish because of great variations in the number of seams in the Christensen Zone, landward pinchouts and interfingering of littoral sandstone beds, and a few uncompleted drill holes (#30 and #31). The reliability of correlations in this area are probably on the order of 70%.

The correlation activities described in the methods section of this report (2.23) has resulted in the definition of three major coal zones (Appendix 2-5) which are comprised of from one to eight coal seams. The average thickness of the Christensen Zone on the Garfield Deposit is 128.4 feet, the Rees Zone averages 87.4 feet, and the Alvey Zone, 30.9 feet (Table 2-5).

TABLE 2-5

STRATIGRAPHIC UNIT THICKNESSES (FEET)

Hole	Alvey Stratigraphic Unit Thickness	Rees Stratigraphic Unit Thickness	Christensen Stratigraphic Unit Thickness
1	61.8	43.4	117.5
2	65.8	72.3	69.1
3	22.1	72.3	61.7
4	34.1	41.6	161.0
5	24.4	61.5	123.4
6	47.3	156.9	110.9
7	0	37.1	161.2
8	31.4	130.0	95.2
9	18.1	82.6	125.4
11	30.9	47.2	64.0
12	47.3	141.6	97.9
20	22.6	3.3	129.2
21	79.9	43.1	100.5
22	55.1	26.7	114.5
23	47.3	59.4	141.5
24	38.0	63.7	137.9
25	23.1	121.6	100.3
26	49.7	142.8	169.7
27	24.2	108.6	158.9
28	6.2	142.0	192.8
29	6.8	92.5	190.5
30	4.0	146.1	82.8
31	0	129.8	166.4
32	2.5	131.1	209.4
Mean	30.94	87.38	128.4
1st SD	22.33	45.60	41.44

TABLE 3-5

STRATIGRAPHIC UNIT THICKNESS (FEET)

Hole	Ninety Stratigraphic Unit Thickness	Mass Stratigraphic Unit Thickness	Christensen Stratigraphic Unit Thickness
1	61.8	43.4	117.5
2	86.3	72.3	69.1
3	22.1	72.3	81.7
4	34.1	81.5	181.0
5	24.0	81.5	133.4
6	47.3	156.3	110.3
7	0	37.1	161.2
8	31.4	130.0	86.2
9	18.1	82.6	125.4
11	30.9	82.7	84.0
12	44.3	141.6	87.9
26	22.6	3.3	128.3
27	74.3	83.1	100.5
22	34.1	38.7	114.5
23	47.3	38.6	161.5
24	38.0	83.7	137.8
25	23.1	121.6	100.3
28	88.7	102.8	169.7
29	24.2	108.6	138.0
30	8.2	142.0	152.8
31	0.8	82.5	190.5
32	4.8	106.1	82.8
33	0	120.8	156.4
37	2.5	121.1	209.4
Mean	30.34	87.38	126.4
Std Dev	22.13	45.80	41.44

The nomenclature that has been developed to identify the fifteen coal seams within the deposit is based on a numerical/alphabetical heirarchy (Figure 2-9). The name of each coal seam is prefixed by one of the letters "C", "R", or "A" which correspond to the Christensen, Rees, and Alvey Coal Zones, respectively. The next symbol behind the zone prefix is numerical which refers to the position of a seam relative to the base of each zone. In other words, the basal seam of the Rees Zone is the R1, the uppermost seam in that zone is the R6. Rock splits that subdivide major seams into sub-seams necessitates the use of an alphabetical suffix for naming the sub-seam. The basal sub-seam is then an "A" sub-seam and the upper sub-seam a "B" sub-seam. For instance, the R1 seam splits into the R1A (lower) and R1B (upper) sub-seams. Further subdivisions alternate between numerical and alphabetical suffixes with the basal unit always the lowest number or start of the alphabet. The finest subdivision of coal seams in the Garfield Deposit are the R1A2 and C3B2 sub-seams (Figure 2-9).

The nomenclature that has been developed for the major seams within the Garfield Deposit has been noted on the fence diagrams (Appendix 2-5), cross sections (Appendix 2-2), as well as on the Coal Summaries (Appendix 2-6).

Distribution of Significant Units

The nomenclature/correlation diagram shown in Figure 2-9 summarizes the general development of each of the major coal seams within the deposit as well as the stratigraphic position of the major littoral sandstone bodies that underlie each of the major coal zones. This diagram has been reproduced in the lower left-hand corner of each of the maps that have been prepared to illustrate the distribution of coal seams and interburden between seams, the overburden over coal seams, and the structure of the seams. The variations in thickness of each of the seams in this diagram is representative,

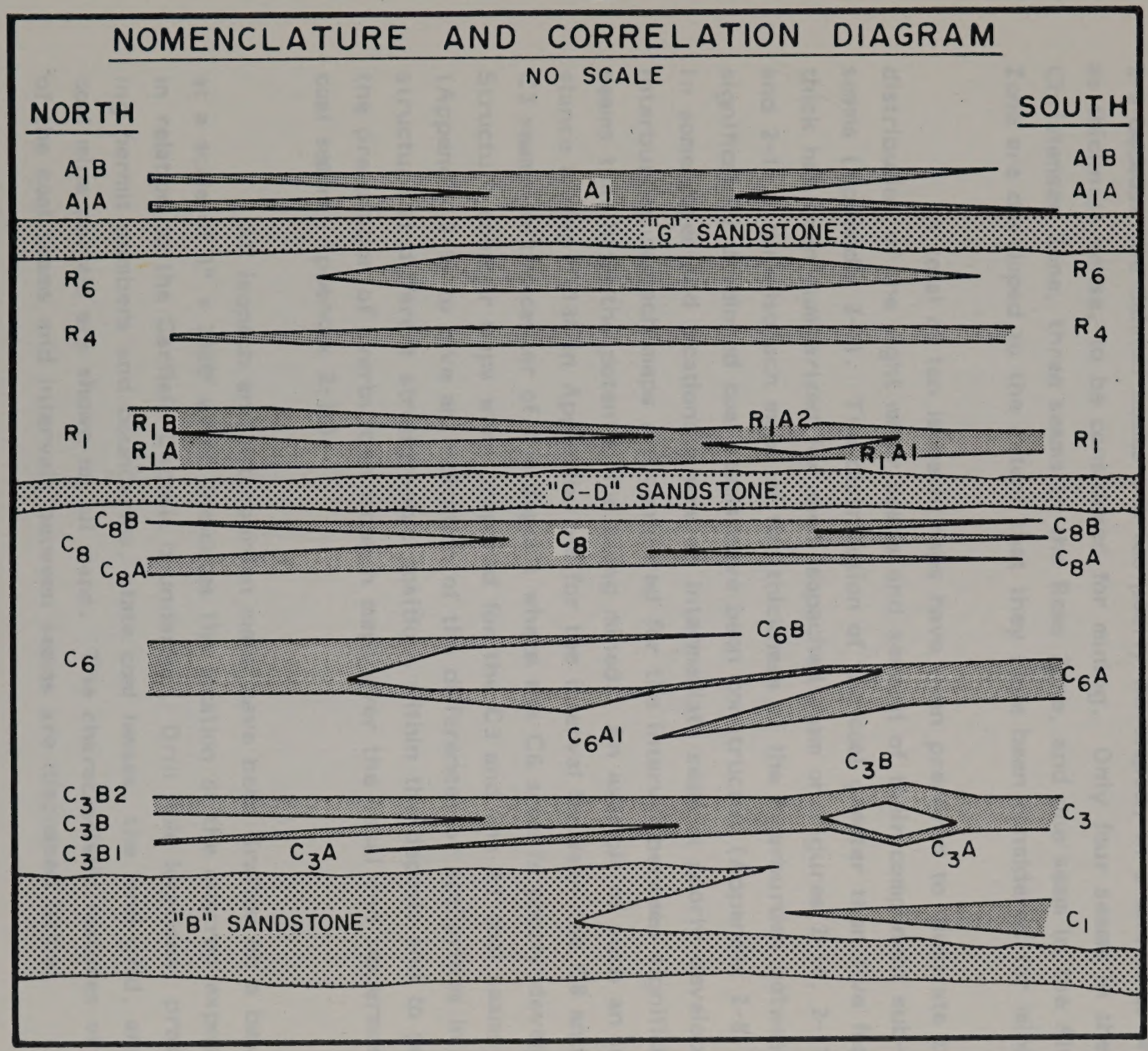
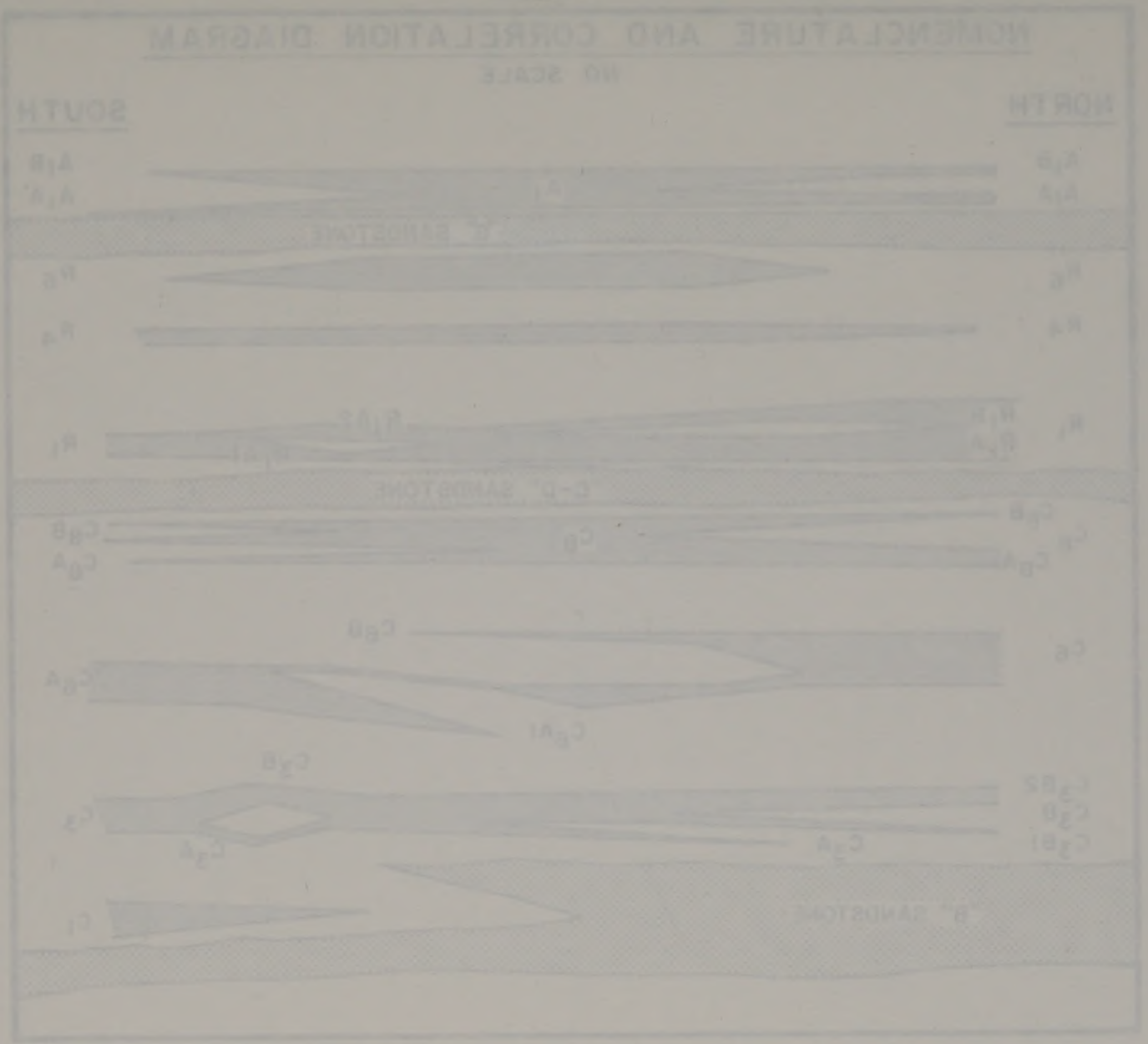


FIGURE 2-9: Correlation and Nomenclature Diagram for the Major Coal Seams of the Garfield Deposit. Major Widespread Littoral Sandstone Beds Shown for Reference.

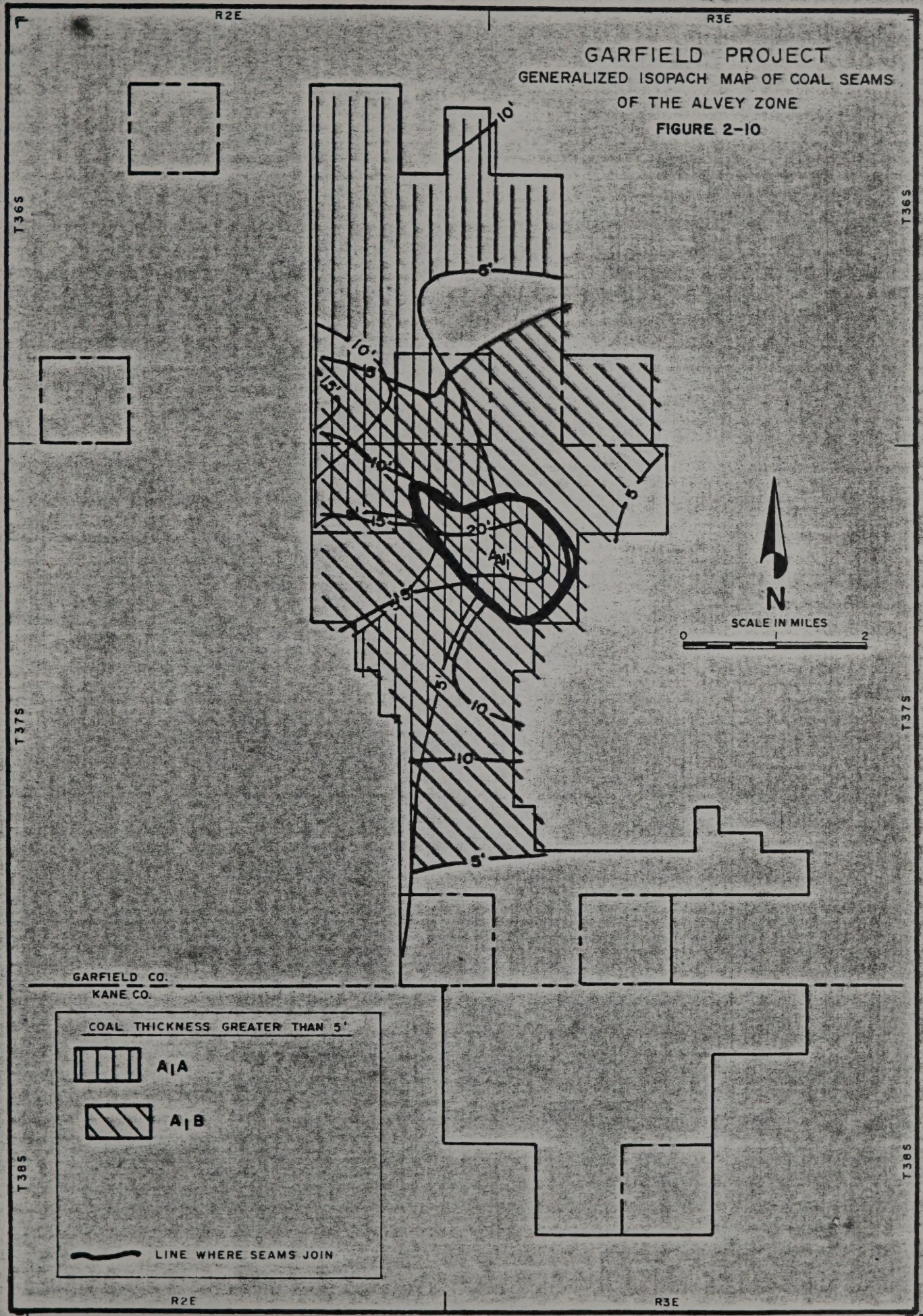


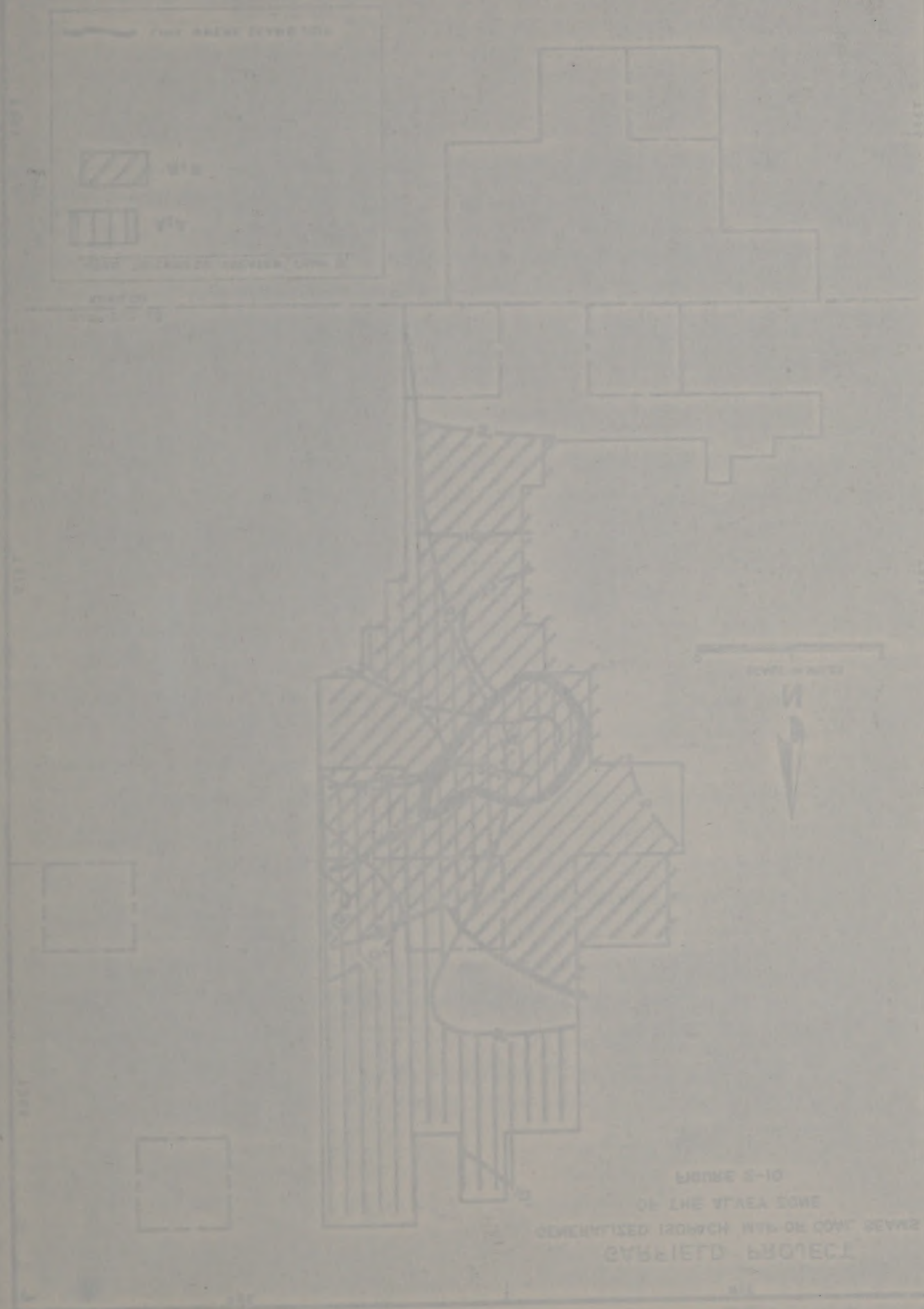
in a general sense, of the degree of development of each of the seams in a north-south direction within the deposit.

Not all of the fifteen named seams have been illustrated on Figure 2-9 because the deleted seams are too poorly developed, in a lateral as well as thickness sense, to be considered for mining. Only four seams in the Christensen Zone, three seams in the Rees Zone, and one seam in the Alvey Zone are developed to the extent that they have been considered for mining.

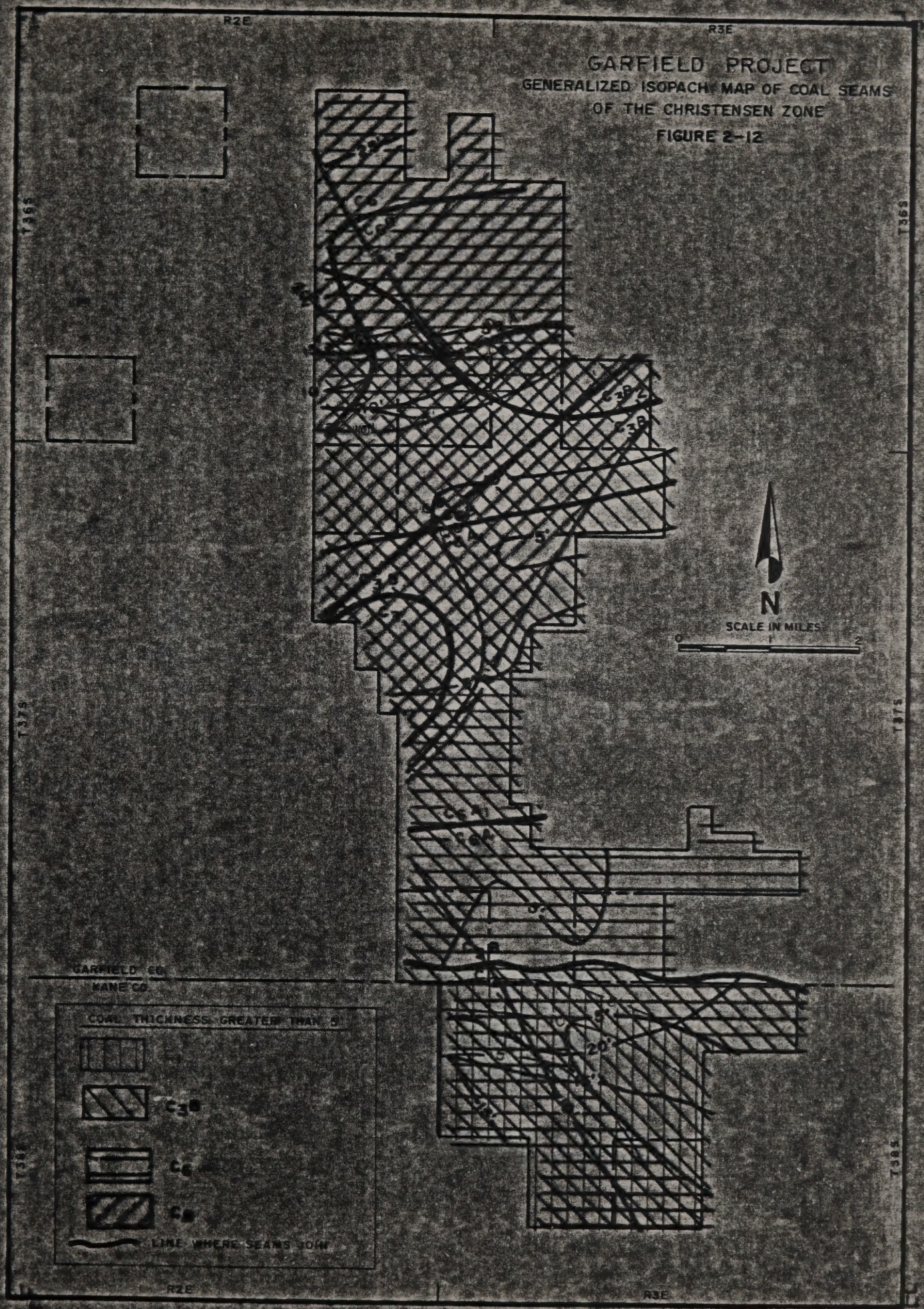
A total of ten isopach maps have been prepared to illustrate the distribution of the eight major seams and several of their component sub-seams (Appendix 2-7). The distribution of the coal greater than five feet thick has been summarized for each isopached seam on Figures 2-10, 2-11, and 2-12. Nine isopach maps of the thickness of the interburden between significantly developed coal seams have been constructed (Appendix 2-8). In some cases and locations where an intermediate seam is poorly developed, interburden isopach maps are contoured for the interval between significant seams that have the potential for being mined. An example of such an instance is Map 10184 in Appendix 2-8 for the interval between the C8 and C3 seams in the center of the deposit where the C6 seam is poorly developed. Structure contour maps were prepared for the C3 and the A1 coal seams (Appendix 2-9) to give an indication of the differences or similarities in the structure at different stratigraphic positions within the deposit and to permit the preparation of overburden isopach maps over the basal and uppermost coal seams (Appendix 2-10).

All isopach and interburden maps have been drafted on a base at a scale of 1" = 2000' which indicates the location of the outcrop exposures in relation to the Garfield Deposit boundaries. Drill hole locations, prospecting permit numbers and boundaries, state coal leases, the land grid, and coal mine portals are shown on all maps. The characteristic features of each of the coal seams and intervals between seams are discussed below.


R2E
R3E
GARFIELD PROJECT
GENERALIZED ISOPACH MAP OF COAL SEAMS
OF THE ALVEY ZONE
FIGURE 2-10
T36S
T37S
T38S
10'
5'
10'
5'
15'
10'
5'
15'
20'
A1
15'
5'
10'
10'
5'
N
SCALE IN MILES
0
2
GARFIELD CO.
KANE CO.
COAL THICKNESS GREATER THAN 5'
A1A
A1B
LINE WHERE SEAMS JOIN
R2E
R3E



R2E
R3E
GARFIELD PROJECT
GENERALIZED ISOPACH MAP OF COAL SEAMS
OF THE REES ZONE
FIGURE 2-11
T36S
T37S
T38S
N
SCALE IN MILES
0
1
2
R1A
R1
R1A
R1
GARFIELD CO.
KANE CO.
COAL THICKNESS GREATER THAN 5'
R1A
R1B
R4
LINE WHERE SEAMS JOIN


R2E
R3E
GARFIELD PROJECT
GENERALIZED ISOPACH MAP OF COAL SEAMS
OF THE CHRISTENSEN ZONE
FIGURE 2-12
T36S
T37S
T38S
N
SCALE IN MILES
0
1
2
GARFIELD CO.
KANE CO.
COAL THICKNESS GREATER THAN 0
C3B
C6
C8
LINE WHERE SEAMS JOIN

C1 Coal Seam

The C1 coal seam is present only in the most southern portion of the property (Figure 2-10). The seam thins to the north and appears to thicken southward (Appendix 2-7; Map 10189).

C1-C3 Interval

The interval between the C1 and C3 seams thickens to the northeast and thins to the southwest (Appendix 2-8; Map 10185). The interval averages about 55 feet thick in areas where it is overlain by coal in excess of five feet thick in the C3 seam.

C3 Coal Seam

The C3 coal seam is unsplit in the most southern portion of the property. The seam splits and the upper sub-seam (C3B) is best developed in the central portion of the property (Figure 2-10). In a small portion in the west central area the C3 sub-seams join and form an unsplit C3 seam. The upper sub-seam (C3B) splits to the north and the C3B2 of the lower sub-seam, is the best developed. The C3 seam is thickest where it is unsplit (Appendix 2-7; Map 10188).

Overburden is thinest in the southern and northern portions of the property and reaches its maximum thickness over the central portion of the property (Appendix 2-10; Map 10169).

The structure of the C3B seam is characterized by a continuous syncline along the western edge of the property and by an anticline to the extreme southeast (Appendix 2-9; Map 10171). Both structures are very subtle and conform to the structure maps prepared by Zeller (1972a through d).

C3-C6 Interval

The interval between the C3 and C6 coal seams is thickest in the southern end of the property and thickens to the northwest. In the northern portion, the interval thickens to the northeast and thins to the southwest (Appendix 2-8; Map 10186).

C6 Coal Seam

The C6 coal seam is unsplit in the extreme northwestern portion of the property. The C6 splits and the C6B thins to the central portion, where the C6A1 becomes well developed and thickens to the south until the C6A joins (Figure 2-10). The C6A generally thickens southward with a northwestward thin zone bisecting it (Appendix 2-7; Map 10187).

C6-C8 Interval

The C6-C8 interval is only applicable to a small portion in the central part of the property and the north part of the property. In the central portion the interval shows a southwestward thickening and to the north the interval thins to the north and south to a belt bisecting it (Appendix 2-8; Map 10184).

C8 Coal Seam

The C8 coal seam is unsplit in the north-central portion of the property (Figure 2-10). It splits and the C8B thins rapidly southeastward. To the north the C8 splits and the C8A is well developed. The C8 shows a northeasterly thickening in the north-central portion of the property (Appendix 2-7; Map 10190).

C8-R1 Interval

The C8-R1 interval roughly shows an isopach of the C-D sandstones. The interval thins to the southwest and thickens to the northeast (Appendix 2-8; Map 10191).

R1A Coal Seam

The R1 seam is unsplit in the extreme northwest corner of the property and splits going to the south where the R1A is best developed (Figure 2-11). The R1A thickens southward and rejoins the R1 in a small pod in the eastern central portion of the property. The R1A splits to the south and the R1A2 is better developed. In the southeastern portion, the R1A2 joins and the R1A is unsplit. The R1A then thins to the southeast (Appendix 2-7; Map 10178).

R1A-R1B Interval

The R1A-R1B interval reaches it maximum thickness in two northeastward to eastward trending belts. The interval thins to the north and south (Appendix 2-8; Map 10181).

R1B Coal Seam

The R1B coal seam is present in the northwest portion of the property and consists of two pods which trend eastward or southeastward (Figure 2-11; Appendix 2-7; Map 10177).

R1A and R1B-R4 Interval

The R1-R4 interval in the southern portion of the property thickens to the west. In the central portion several thick pods are present. While to the north an eastward trending thick zone is present which may represent a fluvial channel (Appendix 2-8; Map 10183).

R4 Coal Seam

The R4 coal seam is unsplit over the entire property (Figure 2-11). The R4 consists of one large thick coal pod in the center of the property and a smaller one in the southern portion of the property (Appendix 2-7; Map 10179).

R4-R6 Interval

The R4-R6 interval is only present in the southern half of the property. The interval shows a general westward thickening with a very thick pod present in the extreme southeastern portion of the deposit (Appendix 2-8; Map 10182).

R6 Coal Seam

The R6 is unsplit and is present in only the southern half of the deposit (Figure 2-11). The R6 consists of two pods; a well developed one in the central portion of the deposit and the other in the southernmost portion of the deposits (Appendix 2-7; Map 10180).

R6-A1A Interval

The R6-A1A interval is present in the southern half of the property and shows a southwestward thinning (Appendix 2-8; Map 10176).

R1A Coal Seam

The R1A coal seam is the lower sub-seam of the A1 and is present only in the northern portion of the deposit (Figure 2-12). It consists of a pod which thins to the north, south, and east (Appendix 2-7; Map 10173).

A1A-A1B Interval

The A1A-A1B interval is zero in the central portion of the property where A1A and A1B join. The interval generally thickens away from the center pod except in the extreme northern portion where it thins northward (Appendix 2-8; Map 10175).

A1B Coal Seam

The A1B coal seam is the upper sub-seam of the A1 and exhibits a southward and northward thinning (Figure 2-12; Appendix 2-7; Map 10174). The overburden is thickest in the central portion of the property and generally thins to the north and south (Appendix 2-10; Map 10170). The structure is characterized by a dominant syncline along the western edge of the property and by a smaller anticline in the northern part of the property. Both structures are subtle (Appendix 2-9; Map 10168).

Summary of Seam Distributions

The distribution of the coal seams and the intervals between seams indicates a great similarity with the continuities for seams and interburden intervals for the Garfield Deposit as compared to other coal deposits that have been drilled on close spacing patterns in the Kaiparowits Plateau. As is characteristic for those deposits, coal seams can be correlated in terms of tens of square miles and seams as well as interburden thicknesses can be quite lenticular.

Generalized data from the 24 completed drill holes indicates that the average seam thickness for the seams and sub-seams isopached is about 9.6 feet and that the average interval between seams is about 47 feet (Table 2-6).

TABLE 2-6

MAP DATA TABULATION

Appendix Number	UP&L/M&E Map Number	Description	Average Coal Thickness*	Generalized Avg. Interburden Thickness
2-7	10189	C1 Coal Seam	13.2	
	10188	C3A Coal Seam	11.5	
	10188	C3B Coal Seam		
	10187	C6A1 Coal Seam		
	10187	C6A2 Coal Seam	10.6	
	10187	C6B Coal Seam		
	10190	C8 Coal Seam	11.4	
	10178	R1A Coal Seam	7.2	
	10177	R1B Coal Seam	6.9	
	10179	R4 Coal Seam	7.6	
	10180	R6 Coal Seam	8.4	
	10173	A1A Coal Seam	8.0	
	10174	A1B Coal Seam	11.0	
2-8	10175	A1B-A1A Interval		25
	10176	A1A-R6 Interval		64
	10182	R6-R4 Interval		62
	10183	R4-R1B Interval		33
	10181	R1B-R1A Interval		10
	10191	R1A-C8 Interval		100
	10184	C8-C6B Interval		38
	10186	C6A1-C3B Interval		35
	10185	C3A-C1 Interval		55
2-9	10168	Structure Contour A1B		
	10171	Structure Contour C3		
2-10	10170	Overburden A1B		
	10169	Overburden C3		
			$\bar{X}$ = 9.58	$\bar{X}$ = 46.89
			S = 2.21	S = 26.60

*For seams greater than 5 feet

TABLE 2-6

MAP DATA TABULATION

Appendix Number	USGS/MGE Map Number	Description	Average Coal Thickness *	Generalized Avg. Interburden Thickness
2-7	10159	C1 Coal Seam	13.2	
	10163	C3A Coal Seam	11.3	
	10162	C3B Coal Seam		
	10167	CBA1 Coal Seam		
	10167	CBA2 Coal Seam	10.8	
	10167	CBB Coal Seam		
	10169	C8 Coal Seam	11.4	
	10174	R1A Coal Seam	7.2	
	10177	R1B Coal Seam	8.9	
	10179	R4 Coal Seam	7.6	
	10180	R6 Coal Seam	8.4	
	10173	A1A Coal Seam	8.0	
	10179	A1B Coal Seam	11.0	
2-8	10175	A1B-A1A Interval		35
	10176	A1A-R6 Interval		86
	10182	R6-R4 Interval		62
	10183	R4-R1B Interval		33
	10181	R1B-R1A Interval		10
	10197	R1A-C8 Interval		100
	10193	C8-C3B Interval		38
	10186	CBA1-C3B Interval		35
	10165	C3A-C1 Interval		55
2-9	10166	Structure Contour A1B		
	10171	Structure Contour C3		
2-10	10176	Overburden A1B		
	10165	Overburden C3		

$\bar{X}$ = 9.56 $\bar{X}$ = 56.59

S = 1.21 S = 26.60

*For seams greater than 5 feet

Reserves in the eight seams that were isopached were calculated assuming an in-place density of 1800 tons/acre-foot for coal seams greater than five feet in thickness. Although published sources sometimes indicate that sub-bituminous coals, the rank of the coals of the Garfield Deposit, have a specific gravity of 1.30 and weigh 1770 tons/acre-foot, available density information from 161 samples tested by Commercial Testing and Engineering Company. (CT and E) indicates that the average specific gravity of the Garfield coals is 1.44 or in excess of 1950 tons/acre-foot. UP&L has taken a conservative approach to calculating reserves more in keeping with conventional density factors and has chosen a weight factor of 1800 tons/acre-foot for reserve calculations.

Reserves were calculated by planimeter methods where all isopached coal seams greater than five feet in thickness were planimetered to the nearest one-hundredth of a square inch and converted to acres. Acreages were then multiplied by average seam thicknesses in either one and two foot increments (depending on the seam) and multiplied by 1800 tons/acre-foot. The reserves for each of the isopached coal seams is shown on the coal isopach maps and is tabulated in Table 2-7. A total of 1.499 billion tons of in-place coal reserves have been calculated for the deposit, of which 1.286 billion (86%) is federal and 212 million (14%) is state. Reserves were not calculated for the two state sections to the west of the contiguous block of lands inasmuch as no drill hole data are available in that area. Additional coal reserves at thicknesses greater than 5' are present in some of the minor intermediate seams between the major seams that have been isopached (C2 or C7, for instance), but the thinness of the interval between these seams and surrounding seams is too small to consider them of mineable significance and so reserves have not been calculated for them.

The isopach maps and reserve calculations indicate that a significant portion of the Garfield Deposit is comprised of coal seams that are in

TABLE 2-7

IN-PLACE COAL RESERVES ON GARFIELD DEPOSIT

(Tons)

Seam	Federal	State	Total
A1B	139,189,500	27,784,400	166,977,900
A1A	78,392,700	8,531,000	86,923,700
R6	48,419,100	-	48,419,100
R4	127,375,200	17,913,600	145,288,800
R1B	47,216,700	8,831,700	56,048,400
R1, R1A, R1A2	102,184,200	19,620,900	121,805,100
C8	182,102,400	35,914,600	218,017,000
C6	211,320,000	28,334,700	239,654,700
C3B	241,371,000	46,237,500	287,608,500
C1	109,059,300	19,534,500	128,593,800
	1,286,630,100	212,706,900	1,499,337,000

TABLE 2-7

IN-PLACE COAL RESERVES ON CARFIELD DEPOSIT

(Tons)

Seam	Federal	State	Total
A1B	139,189,500	27,769,400	166,977,900
A1A	78,382,700	8,831,000	86,923,700
R6	46,619,700	-	46,619,100
R4	127,375,200	37,913,600	165,288,800
R1B	47,219,700	8,831,700	56,050,400
R1, R1A, R1A2	102,198,200	19,020,900	121,805,100
C6	182,102,400	35,819,600	218,077,000
C4	211,320,000	28,339,700	239,659,700
C3B	241,371,000	46,237,500	287,608,500
C1	109,059,300	19,539,500	128,593,800
	1,288,630,100	213,706,900	1,499,337,000

TABLE 2-7

IN-PLACE COAL RESERVES ON GARFIELD DEPOSIT

(Tons)

Seam	Federal	State	Total
A1B	139,189,500	27,784,400	166,977,900
A1A	78,392,700	8,531,000	86,923,700
R6	48,419,100	-	48,419,100
R4	127,375,200	17,913,600	145,288,800
R1B	47,216,700	8,831,700	56,048,400
R1, R1A, R1A2	102,184,200	19,620,900	121,805,100
C8	182,102,400	35,914,600	218,017,000
C6	211,320,000	28,334,700	239,654,700
C3B	241,371,000	46,237,500	287,608,500
C1	109,059,300	19,534,500	128,593,800
	1,286,630,100	212,706,900	1,499,337,000

excess of ten feet in thickness. The reserve calculations presented on each of the isopach maps have been reworked to determine the quantity of reserves in the increment of ten feet thick and greater (Table 2-8). These data indicate that 17.1% of the federal coal reserves, and 11.5% of the state reserves, are within this category. The reserves in the increment between five and ten feet in thickness, regardless of total seam thickness, amounts to 1.036 billion tons of coal on the federal lands.

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UP&L classifies the reserves of the Garfield Deposit as within the proven and indicated categories. The experiences of other companies that have completed drilling programs on one-half mile center (or closer) within the Kaiparowits Plateau (El Paso Natural Gas Co. and the consortium of utilities involved in the Kaiparowits Region) indicate that this classification can be substantiated because of the lateral continuity of coal seams and great similarities of geology for all deposits in the plateau. The experiences of these companies who proceeded from wide-spaced drilling patterns of about one mile centers to the tight spacing patterns indicated that the generalities of seam correlations can be defined to the extent that the reserves can be estimated within a ± 10% degree of accuracy with drill holes on one mile centers. Close spaced drilling patterns refine the correlations of individual seams but the magnitude of the reserve remains essentially the same.

The methods of defining measured, indicated, and inferred reserves proposed in U.S.G.S. Bulletin 1450-B are not directly applicable to the coal reserves of the Garfield Deposit inasmuch as the continuity of coal seams has been demonstrated to be well in excess of the tight restrictions imposed by Bulletin 1450-B. If reserves were calculated in strict accordance with the Bulletin using various radii of influence around drill holes, only 65% of the deposit would be included as measured and indicated with the remainder as inferred (Appendix 2-11).

TABLE 2-8

IN-PLACE RESERVES WITHIN THE INCREMENT OF 10' AND GREATER FOR THE GARFIELD DEPOSIT

FEDERAL LANDS

(Thousands of Tons)

Seam Name	Total Reserves	5-10' Thick	>10' Thick	% >10' Thick
A1B	139,190	107,436	31,754	22.8
A1A	78,393	73,880	4,513	5.8
R6	48,419	45,858	2,561	5.3
R4	127,375	124,990	2,385	1.9
R1B	47,217	47,217	-	0
R1, R1A, R1A2	102,184	98,570	3,614	3.5
C8	182,102	149,002	33,100	18.2
C6	211,320	168,350	42,970	20.3
C3B	241,371	180,207	52,164	21.6
C1	109,059	62,042	47,018	43.1
Subtotal	1,286,630	1,066,551	220,079	17.1
		STATE LANDS		
A1B	27,788	24,381	3,407	12.3
A1A	8,531	8,241	290	3.4
R6	-	-	-	-
R4	17,914	17,914	-	-
R1B	8,832	8,832	-	-
R1, R1A, R1A2	19,621	19,049	572	2.9
C8	35,915	34,706	1,209	3.4
C6	28,335	25,755	2,579	9.1
C3B	46,238	34,326	6,912	14.9
C1	19,535	10,022	9,513	48.7
Subtotal	212,707	188,224	24,483	11.5
TOTAL	1,499,337	1,254,775	244,562	16.3

Coal quality information is available from two main sources: (1) 161 core samples collected by UP&L and tested by Commercial Testing and Engineering, and (2) 46 samples collected by Doelling and Graham (1972) from various locations within the four quadrangles in which the deposit is located. The data from Doelling and Graham (1972) is summarized in Table 2-9 below:

Table 2-9
Coal Quality Data from Outcrops
(Doelling and Graham, 1972)

	Number of Samples	Average	Standard Deviation
Ash Content	46	6.87%	3.37%
Moisture Content	46	10.41%	5.78%
BTU Value	28	10,543	1,017

Data compiled by UP&L consists of 161 coal samples from drillhole cores (Appendix 2-12). These data have been summarized in two ways. The first method was to evaluate all coal samples using the five different assumptions listed below:

1. Total evaluation	Running average of all cores with all rock splits, roof, and floor rock included.
2. High ash 5'	For only those seams greater than 5' thick, including rock.
3. Low ash 5'	For only those seams greater than 5' thick, excluding all samples with ash contents 15%; subdivided seams are combined as single samples.
4. High ash 10'	Seams 10' thick with all rock included.
5. Low ash 10'	Seams 10' thick, with samples 15% ash excluded; subdivided seams combined as a single sample.

Coal Quality

Coal quality information is available from two main sources:
(1) 181 core samples collected by UPEL and tested by Commercial Testing and Engineering, and (2) 46 samples collected by Doelling and Graham (1972) from various locations within the four quadrangles in which the deposit is located. The data from Doelling and Graham (1972) is summarized in Table 2-9 below:

Table 2-9
Coal Quality Data from Outcrops
(Doelling and Graham, 1972)

	Number of Samples	Average	Standard Deviation
Ash Content	46	6,878	3,378
Moisture Content	46	10,414	3,788
BTU Value	28	10,582	1,012

Data compiled by UPEL consists of 181 coal samples from drillhole cores (Appendix 2-12). These data have been summarized in two ways. The first method was to evaluate all coal samples using the five different assumptions listed below:

1. Total evaluation — Running average of all cores with all rock splits, roof, and floor rock included.
2. High ash 3' — For only those seams greater than 3' thick, including rock.
3. Low ash 3' — For only those seams greater than 3' thick, excluding all samples with ash contents 15%; subdivided seams are combined as single samples.
4. High ash 10' — Seams 10' thick with all rock included.
5. Low ash 10' — Seams 10' thick, with samples 15% ash excluded; subdivided seams combined as a single sample.

The statistical information that has been derived in accordance with the five assumptions noted above are presented in Table 2-10. The data shown in Row 3 of Table 2-10 are probably representative of the in-place quality of the coals that can be considered for mining.

The second method used to collate the UP&L core samples was to summarize the quality information available concerning each isopached seam (Table 2-11 and Appendix 2-13). This tabulation indicates that the data collated in Row 3 of Table 2-10 is probably the most representative of the coals that have potential for mining.

Additional coal quality data concerning the mineral analysis of ash, as tested by Standard Laboratories, Inc., for 96 of the splits from the 161 core samples collected by UP&L are tabulated in Table 2-12.

The three major collations of datanoted above indicate some discrepancies. The data from Doelling and Graham (1972) indicate a higher BTU vlaue and lower moisture content for the Garfield coals than is indicated in the UP&L core data. The high moisture content and associated decrease in BTU content of the Garfield core samples is probably a function of moisture from drilling fluids that was enclosed with samples analyzed by CT&E and may not be representative of the coal from the deposit. Additional proprietary core data for core samples collected in accordance with more proper sampling procedures from an adjacent property in the Kaiparowits Plateiau for coal seams equivalent to the Christensen zone tend to support a moisture content of about 10% for the Garfield coals. These data will be discussed in a discussion on coal quality dilution in the mine plan section of the report. A more likely value fo BTU's and moisture content is probably comparable to the outcrop samples collected by Doelling and Graham (1972).

In summary, the average coal quality for the isopached coal seams of the Garfield Deposit is presented in the "Average" column of Table 2-13 (for ash, BTU's, and moisture), Table 2-10, Row 3, for other proximate analyses as well as ash fusion temperatures, and Hardgrove grindability, and Table 2-12 for the mineral analysis of ash.

TABLE 2-10

SUMMARY OF COAL QUALITY DATA FOR GARFIELD CORES

	Proximate Analyses											
	Thickness	As Received Basis						Dry Basis				
		Moisture	Ash	BTU's	Volatile	Carbon	Sulfur	Ash	BTU's	Volatile	Carbon	Sulfur
1. All Evaluations	7.5±4.4	15.0±2.2	9.0±6.5	10135±879	36.7	39.4	0.82±.42	10.5±7.1	11913±1031	43.1	46.4	0.97±.48
2. ≥ 5' Seams High Ash Incl.	9.7±4.7	15.2±2.2	8.6±4.1	10168±678	36.7	31.7	0.77±.34	10.0±5.2	11981±812	43.2	46.8	0.90±.39
3. ≥ 5' Seams Excl. ≥15% Ash	9.5±4.4	15.6±2.0	7.2±2.4	10311±505	36.9	40.4	0.72±.29	8.6±2.9	12204±490	43.6	47.8	0.85±.34
4. ≥ 10' Seams High Ash Incl.	15.2±4.4	15.3±2.7	9.4±5.6	10049±703	36.7	39.1	0.66±.20	11.0±6.0	11877±867	42.8	46.2	0.78±.23
5. ≥ 10' Seams Excl. ≥15% Ash	15.2±4.0	15.8±2.6	7.8±2.6	10204±454	37.1	39.8	0.63±.14	9.3±3.0	12118±471	43.4	47.3	0.75±.16

	Ash Fusion Temperatures (degrees Fahrenheit)								Hardgrove Grindability	True Specific Gravity, Dry
	Reducing Conditions				Oxidizing Conditions					
	Init. Defor.	H = W	H = ½W	Fluid	Init. Defor.	H = W	H = ½W	Fluid		
1. All Evaluations	2152± 87	2198±127	2253±154	2357±193	2204± 99	2254±128	2325±148	2438±169	49.3±3.9	1.46±.09
2. ≥ 5' Seams High Ash Incl.	2150± 73	2210±109	2250±133	2340±170	2203± 82	2247±106	2320±133	2427±155	48.8±3.4	1.46±.06
3. ≥ 5' Seams Excl. ≥15% Ash	2137± 66	2188± 94	2221±110	2309±158	2188± 73	2223± 86	2294±118	2404±151	48.6±3.2	1.44±.03
4. ≥ 10' Seams High Ash Incl.	2152± 71	2200±113	2252±142	2319±168	2207± 84	2246±109	2316±127	2430±148	48.6±3.9	1.46±.07
5. ≥ 10' Seams Excl. ≥15% Ash	2135± 63	2171± 97	2219±124	2285±158	2190± 78	2220± 94	2288±114	2409±152	48.1±3.5	1.44±.03

Table 2-11

Summary of Coal Quality for Mineable Seams

Seam	No. of Samples	Moisture	Ash	BTU	Sulfur
A1B	12	16.23%	7.91%	10068	0.65%
A1A	10	14.49	11.13	9866	0.72
R6	5	14.37	7.57	10459	0.67
R4	14	14.90	10.12	9985	0.68
R1B	5	14.98	7.95	10235	0.76
R1A,R1,R1A2	16	15.42	8.26	10135	0.96
C8	12	14.77	6.39	10458	0.73
C6	17	15.63	7.85	10247	0.90
C3B	15	14.83	6.67	10511	0.74
C1	1	12.45	13.42	10040	0.68
Average	107	14,81%	8.73%	10201	0.75%

TABLE 2-12

MINERAL ANALYSIS OF ASH FOR GARFIELD DEPOSIT CORES

Parameter	Mean + Standard Deviation
Silicon Dioxide	48.10 ± 8.80
Aluminum Oxide	15.78 ± 3.02
Iron Oxide	5.59 ± 2.27
Calcium Oxide	13.44 ±10.41
Magnesium Oxide	2.22 ± 0.92
Sodium Oxide	1.48 ± 1.61
Potassium Oxide	0.48 ± 0.34
Titanium Oxide	0.78 ± 0.25
Phosphorous Pentoxide	0.09 ± 0.05
Sulfur Trioxide	10.96 ± 3.99
Si/Al Ratio	3.13 ± 0.67
Fe/Ca Ratio	0.61 ± 0.65
Base/Acid Ratio	0.37 ± 0.17

Note: Analyses performed by Standard Laboratories, Inc. on 96 samples.

Table 2-13

Comparison of Various Coal Quality Data

	Sources of Data				Derived
	A	B	C	D	Average
Number of Samples	46	107	161	137	
Moisture, percent	10.41	14.81	15.57	7.80	10.00
Ash, percent	6.87	8.73	7.23	8.20	8.06
BTU, As Received	10543	10201	10311	11600	10683
BTU, Dry Basis	11768	11974	12212	12581	11871

Notes:

1. Underlined values were used in "derived Average" computations.
2. Moisture values in columns B and C are inaccurate because of inclusion of drilling fluids; the derived moisture value was determined on the basis of the data in columns A and D.
3. Ash values in columns A and B were averaged than recalculated on the basis of a 10.00% moisture content to obtain the value shown in the "Derived Average" column.
4. BTU values in columns A and B were average; the wet values are actual, the dry values are calculated; both "Derived Average" values were derived on the basis of a 10.00% moisture content.

Data Sources:

A = Doelling and Graham (1972) coutcrop data

B = UP&L core sample data for isopached seams only (Table 2-11)

C = UP&L core sample data for all coal seams analyzed (Row 3, Table 2-10)

D = Proprietary data obtained from El Paso Coal Co., from the coal seams on land immediately adjacent to UP&L's Garfield Deposit.

Additional subjective information relative to the quality of coal concerns the methane content and spontaneous combustion. The Garfield coals are not thought to be gassy inasmuch as no concentrations of gas were detected in any of the coal seams. The methane content is probably very similar to the coals of the Wasatch Plateau in central Utah.

The coals of the Garfield Deposit, and the Kaiparowits Plateau in general, do not spontaneously combust to any excessive degree. Coal stockpiles and the portals of wagon mines throughout the area are still unburned after several years of exposure. The potential for spontaneous combustion for the Garfield coals is assumed to be the same as for coals of the Wasatch Plateau of central Utah.

2.5 Geologic Summary

Data from 24 drill holes and outcrop mapping activities have been integrated to permit the calculation of 1.28 billion tons of coal in eight major coal seams on federal lands within the Garfield Deposit. These data indicate that the coal is located within three major coal zones that are comprised of from one to as many as eight coal seams. The correlation of coal zones and seams has been simplified by the presence of laterally continuous regressive littoral sandstone bodies within the coal-bearing portions of the Straight Cliffs Formation.

A total of 23 maps at a scale of 1" = 2000' have been prepared to illustrate the distribution of the eight major coal seams, the intervals between seams, the structure of the deposit, and overburden over the basal and uppermost widespread coal seams. The average thickness for the major coal seams in the areas where they are greater than five feet in thickness is 9.6 feet. The average interval between the major seams is 47 feet. The overburden over the deposit ranges from an average of about 900 feet over the uppermost seam to about 1300 feet over the basal widespread seam. The deposit is located near the center of the Alvey Syncline which is characterized by gentle dips that rarely exceed 5° on the deposit.

The lithology within each of the three coal zones consists primarily of mudstone; interbedded siltstone, mudstone, and sandstone; carbonaceous mudstone; and coal (See plates 2-2 and 2-3). Local fluvial sandstone beds are present. The coal zones are separated by widespread littoral sandstone beds that range from 40 to 100 feet in thickness. Minimal amounts of water may be present within the coal zones and in the littoral sandstone beds which, due to their porosity and permeability, probably experience a minor degree of recharge.

The quality of the coal within the Garfield Deposit has been determined on the basis of analysis of samples from both cores and outcrops. The average coal quality is tabulated below:

Ash	8.06 %
Moisture	10.00 %
BTU	10683
Sulfur	0.72 %

2.6 Future Geologic Activities

Present geologic data permit a gross evaluation of the coal seams relations so that general mining plans can be developed for the Garfield Deposit. A more comprehensive understanding of the variations in seam thicknesses, locations of rock splits, intervals and sediments between seams, and local uncertainties in seam correlations must be developed prior to the preparation of detailed mine plans. Additional drilling and mapping activities are necessary in order to obtain this level of understanding for the deposit.

Mapping of the outcrops in the tributary canyons to Right Hand Collet Canyon and Alvey Wash as well as the Straight Cliffs Escarpment will give further insight into the processes that are responsible for the deposition of the coal seams and enclosing sediments which form the deposit. Emphasis placed on the recognition of depositional "packages" should be of use in understanding the genesis of the deposit and therefore of great use in developing the most efficient mine plan. The projection of depositional "packages" into the subsurface and the recognition of those "packages" in drill cores will reinforce and add confidence to correlations. Outcrop mapping activities should take place prior to or concurrent with the commencement of further drilling programs for the deposit.

Drilling programs conducted on other properties in the Kaiparowits Plateau indicate that a drill spacing pattern of one-half mile centers between drill holes is necessary to develop the geologic understanding for the preparation of detailed mine plans. Accordingly, approximately 140 additional drill holes need to be completed on the Garfield Deposit prior to development. Of the 140 holes, 115 holes are planned for the federal lands and 25 for the state lands. A few additional holes may be required in areas of potential portal sites and problem correlations.

The budget necessary to conduct the outcrop mapping and drilling activities discussed above for both the state and federal lands is estimated at a cost of about four million dollars. A breakdown of these costs (Table 2-14) indicates costs of about $24.00 per foot for drilled footage assuming an average drill hole depth of 1200 feet. The mapping and drilling activities are scheduled to take place over the space of two years.

TABLE 2-14

BUDGET FOR FUTURE GEOLOGIC ACTIVITIES

Cost Item	Budget	% of Total
Direct drilling	1,500,000	37.3
Access, site prep., reclamation	290,000	7.2
Personnel and overheads	900,000	22.4
Services (surveying; coal testing)	215,000	5.3
Office and laboratory equipment	30,000	0.7
Facilities (office, trailers)	255,000	6.3
Field equipment (vehicles; radios; tools)	115,000	2.9
Expendables (operating costs)	250,000	6.2
Legal costs	50,000	1.2
Report preparation	50,000	1.2
Contingency (10%)	365,500	9.1
	4,020,500	99.8%

Approximately 82% of the costs budgeted for additional geologic activities for the Garfield Deposit, or $3.3 MM, are applicable to the federal acreage and the remainder, or $0.7 MM, to the state lands.

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3 MINING PLAN

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3. Mining Plan

Approach 3.1

This mining plan to extract the reserves of the Garfield Deposit has been developed in accordance with existing technologies and practices utilized in the coal mining industry and represents a realistic approach to mining a multi-seam coal deposit. The plan has been prepared to be in compliance with existing federal and state regulations and takes into account environmental and reclamation considerations.

Anticipated Conditions 3.11

An interplay of underground and surface conditions determines the potential for success of any mine plan. Underground conditions pertains to the geology and geometry of the deposit. Surface conditions such as topography, geography, and access must be taken into consideration in mine design.

Subsurface conditions: one of the major factors that determines the layout of a mine is the geologic characteristics of the coal deposit. Geologic factors such as hydrology, ground conditions, seam and interburden geometries, stratigraphy, and structure must be taken into account in developing a mine plan that allows the most efficient extraction of coal. The geologic information summarized in Section 2 indicates the presence of a relatively flat-lying coal deposit that consists of three major coal zones comprised of from one to four coal seams each, that are of sufficient thickness and lateral continuity to be considered for mining.

Each coal zone is comprised of several coal seams that are enclosed in mudstones, interbeds, and local sandstones. Each zone is separated from the next by 100 foot thick laterally continuous massive sandstone beds that

3. Mining Plan

3.1 Approach

This mining plan to extract the reserves of the Carfield Deposit has been developed in accordance with existing technologies and practices utilized in the coal mining industry and represents a realistic approach to mining a multi-seam coal deposit. The plan has been prepared to be in compliance with existing federal and state regulations and takes into account environmental and reclamation considerations.

3.11 Anticipated Conditions

An interplay of underground and surface conditions determines the potential for success of any mine plan. Underground conditions pertains to the geology and geometry of the deposit. Surface conditions such as topography, geography, and access must be taken into consideration in mine design.

Subsurface conditions, one of the major factors that determines the layout of a mine is the geologic characteristics of the coal deposit. Geologic factors such as hydrology, ground conditions, seam and interburden geometries, stratigraphy, and structure must be taken into account in developing a mine plan that allows the most efficient extraction of coal. The geologic information summarized in Section 2 indicates the presence of a relatively flat-lying coal deposit that consists of three major coal zones comprised of from one to four coal seams each, that are of sufficient thickness and lateral continuity to be considered for mining.

Each coal zone is comprised of several coal seams that are enclosed in mudstones, interbeds, and local sandstones. Each zone is separated from the next by 100 feet thick laterally continuous massive sandstone beds that

are thought to have aquifer characteristics but are probably not filled with significant quantities of water.

The ten coal seams that have been considered for mining range in thickness from less than 5' to as much as 25'. No one seam has a thickness of over five feet throughout the deposit but several of the seams are of at least five feet thick over several sections of the deposit. Several of the major coal seams are subdivided into sub-seams by rock splits. Coal seams, for the most part, are relatively free of rock splits.

The various pods of coal of mineable thickness (generally greater than 5' thick) in different seams generally overlap but many instances exist where coal pods are isolated from the main block of reserves and main access routes. A fairly intricate system of ramps, slopes, and shafts will be necessary to provide access and ventilation to these pods.

The interburden between mineable seams consists of mudstone, interbeds, and local sandstone beds. Many seams are separated by as much as 100' of strata. Coal seam roof materials are usually mudstone beds and interbeds within the coal zones and sandstone beds for the uppermost seams in each zone. The floor for each of the seams is usually mudstone. See Plates 3-1 and 3-2.

Available data does not indicate the presence of gas in the coals of the Garfield Deposit although minor gas has been encountered in a few of the porous sandstones between the coal zones.

Overburden over the shallowest seam averages about 900' and 1300' over the deepest seam. The thickest overburden over the basal seam is about 1800'.

Surface conditions: the topography of the deposit is characterized by flat-topped mesas and incised canyons. For the most part, surface access routes, mine portals, overland conveyors, and surface facilities can be situated

to take advantage of topographic conditions.

The ridges between incised canyons are the locations where the overburden exceeds 1500' over the basal coal seam. The few deeply incised canyons in the southern part of the deposit provide potential portal sites and air intakes, and minimize shaft sinking depths, but would involve difficult space restrictions if used for main portals (coal haulage belts, mine-mouth preparation plants, and supply yards).

Existing access roads cross through the central part of the deposit. The sparseness of the vegetation and generally inhospitable climate in the area has restricted the influx of humans into the deposit area. Only a few cattle are grazed and no humans have taken up a residence within several miles of the proposed coal mining area.

Mining Method Selection 3.12

Two major methods of mining large volumes of coal have been proven technically and economically feasible: longwall and room-and-pillar methods. The longwall method maximizes recovery, productivity, and safety and minimizes the manpower requirements, environmental disturbance, and some aspects of costs and has been chosen as the major method for extracting the reserves of the Garfield Deposit. This method utilizes room-and-pillar techniques to develop entries to gain access to longwall panels.

The combination of longwall and room-and-pillar mining techniques assumes a minimum mining thickness of five feet (with local four foot coal extraction), a maximum thickness of ten feet, and a minimum separation between seams of thirty feet.

Although a significant tonnage of coal is present in seams greater than ten feet in thickness, recovery of this coal has not proven to be economically viable (though technically feasible) in instances where it has been

PLATE 3-2.

Portal of Alvey Coal Mine.

Atypical sandstone roof on the coal seam
in this location, 1 mile north of the Garfield Deposit.

evaluated. The mining plan proposed here is in accordance with established practices and is assumed to be realistic and conservative in approach.

Longwall panels are designed to be 500 feet wide and up to 5,000 feet long. Continuous miner entries vary in pattern or number, in spacing, and in function, but are generally 20 feet in width, with 50 to 80 feet between centers, and with square pillars in mains and submains and rectangular pillars in panel entries.

The advantage of a longwall system is the higher extraction ratio obtainable, ranging up to 75 or 80 percent of the total recoverable reserve within a given mining area or panel. Use of this technique also results in higher tonnages produced per worker. The continuous miner sections and their role in entry development are viewed as ancillary to the longwall process. Further, longwall panels are easier to schedule because of the higher consistency in their production levels. Downtime from lack of supplies is reduced through a reduction in required logistical support.

Recent installations of longwall equipment in the U.S. have been highly successful due to the inherent increasing reliability and production performance following the traditional room-and-pillar methods. The history of longwall mining in Utah has been long and largely successful as evidenced by the many installations in the Book Cliffs and Wasatch Plateau coal fields of central Utah.

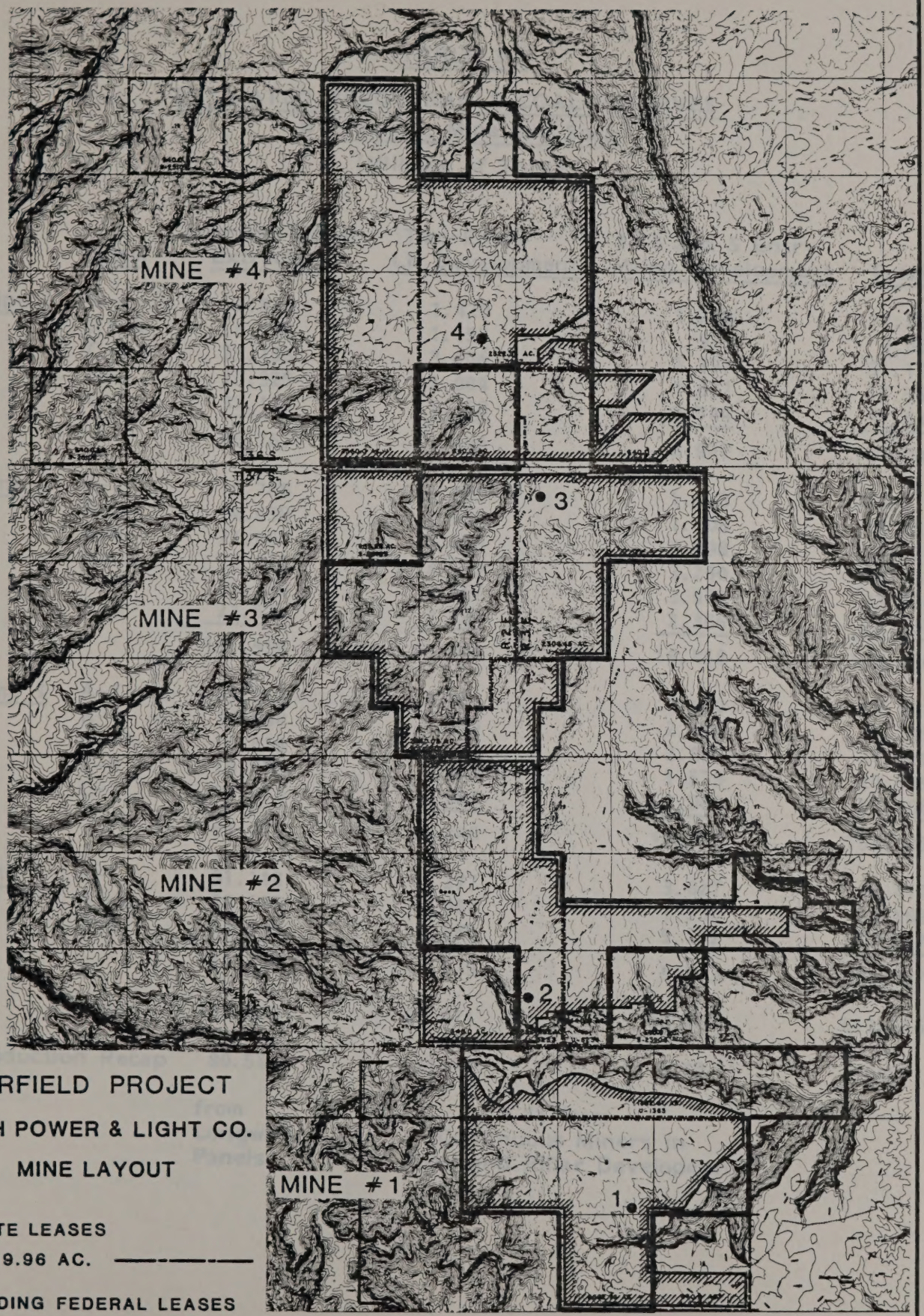
Longwall mining is ideally suited to the anticipated roof conditions of thin bedded mudstones and interbeds with local sandstones which will prefracture and fall as desired without hanging up.

Evaluation of the entire Garfield Deposit geology and lease configuration resulted in a subdivision of the deposit into four "Logical Mining Units". These were established by coal seam continuity and lease shape, with criteria such as minimizing length of entries and maximizing longwall panel layout effectiveness also being considered. The LMU's are illustrated on Figure 3-1.

Selection of mineable seams and layout of mineable areas were done with no scheduling or production capacity demand, other than the general knowledge that the mines would be large, multi-million tons per year in size. Within each mineable seam area, the dimensions of mined areas were measured by length of workings, and multiplied by recovery factors developed for each type of entry pattern and for the longwall panels, at varying seam heights between five and ten feet. This product, the estimated recoverable coal from each seam, is tabulated by mine in Table 3-1.

Note that no coal was projected to be mined in seam heights greater than ten feet, even though such technology may be in standard use by the time this project is underway. Additionally, no recoverable reserve estimate was made for the small areas near outcrops or in areas too small to set up longwall panels, even though that coal will almost certainly be mined. Another siginificant reserve omission is the tonnage likely to be added because of isolation or abandonment of adjacent coal reserves not included within the presently defined leases. These parcels are numerous on the eastern edge of the lease area and are not of interest to other potential developers. These omissions are intentional, and are considered to make the included recoverable coal reserves a very conservative estimate.

Following the calculation of recoverable coal reserves by method, by mine, and by seam, these same mineable areas were tabulated by lessor - State and Federal owned lands - in Table 3-2. About 86% of the Garfield Deposit is Federally controlled.



GARFIELD PROJECT

UTAH POWER & LIGHT CO.

MINE LAYOUT

STATE LEASES
5,119.96 AC. ———-———-——

PENDING FEDERAL LEASES
18,325.16 AC. ———————

FIGURE 3-1

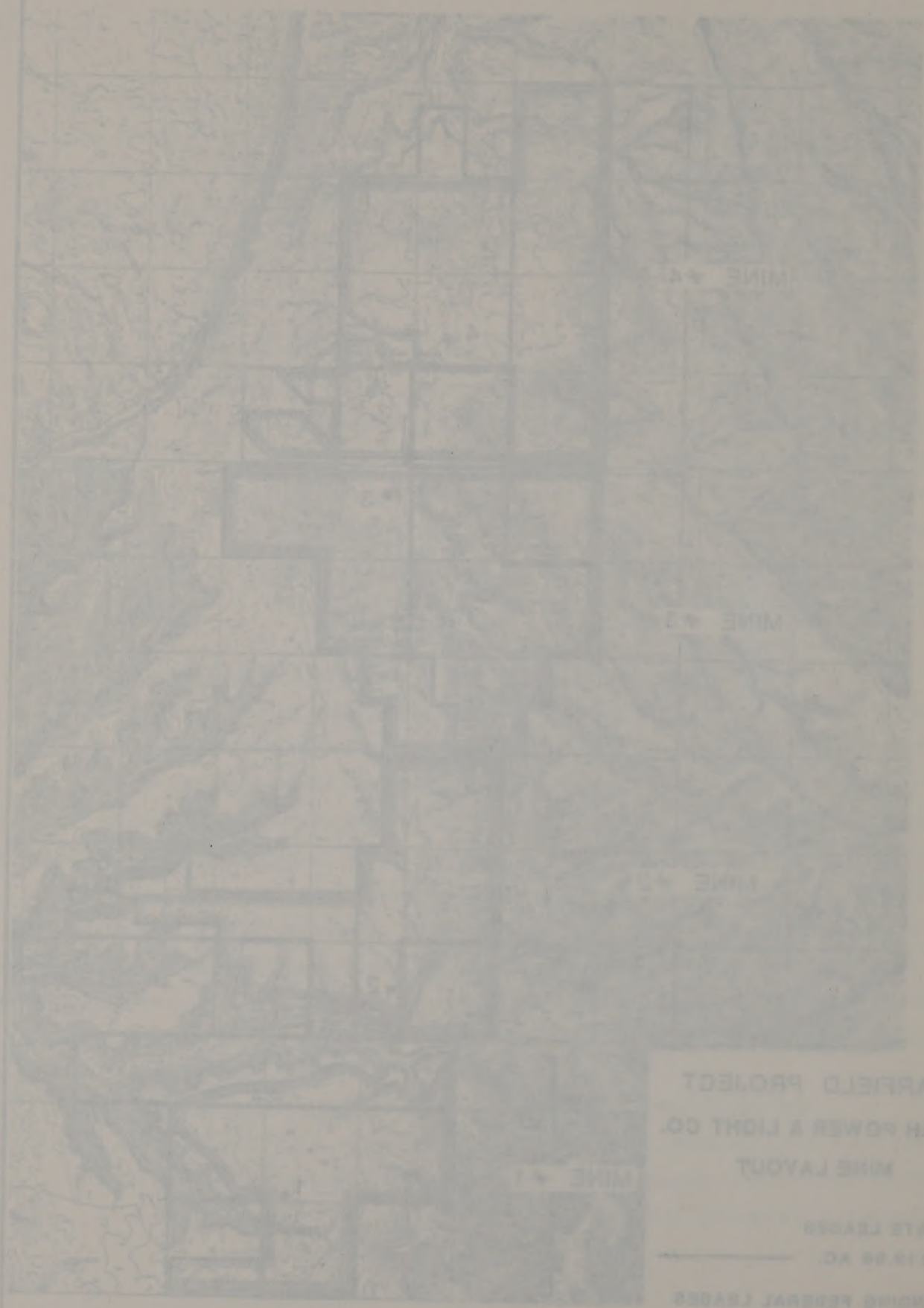


TABLE 3-1

ESTIMATED RECOVERABLE COAL RESERVES

By Method, Mine and Seam

(Millions of Tons)

	Seam	Longwall Panel Extraction	Two-Entry LW Development	3, 4 and 5 Entry Mains Development	Total
Mine 1	R-4	9.579	1.420	1.255	12.254
	R1-A	6.990	1.152	919	9.061
	C-6	25.153	3.646	2.397	31.196
	C-3	13.925	2.004	1.632	17.561
	C-1	31.120	3.768	2.744	37.632
		86.767	11.990	8.947	107.704
Mine 2	A-2	16.786	2.355	1.828	20.970
	R-6	14.638	1.934	1.609	18.181
	R-4	27.467	3.956	3.698	35.122
	R1-A	23.842	3.599	3.290	30.731
	C-6	26.290	3.667	3.667	33.624
	C-3	22.571	3.295	2.366	28.232
		131.594	18.806	16.458	166.860
Mine 3	A-2	48.217	7.154	4.080	59.451
	R-4	30.994	4.585	2.680	38.259
	R1-A	16.816	2.615	2.097	21.528
	C-8	19.978	3.042	1.529	24.549
	C-3	45.973	5.950	4.086	56.008
		161.978	23.346	14.086	199.795
Mine 4	A-1	31.114	4.277	3.027	38.417
	R1-B	23.165	3.468	1.530	28.163
	C-8	58.075	7.985	5.217	71.277
		112.354	15.729	9.773	137.857
TOTAL		492.693	69.871	49.650	612.216
Production Recap		80.5%	11.4%	8.1%	100%
		from Longwall Panels	19.5% from Continuous Miners in Room and Pillar Development		

TABLE 3-1

ESTIMATED RECOVERABLE COAL RESERVES
BY Method, Mine and Seam
(Millions of Tons)

	Seam	Longwall Panel Extraction	Two-Entry LW Development	3, 4 and 5 Entry Mains Development	Total
Mine 1	B-4	8.279	1.420	1.455	12.259
	B1-A	5.886	1.121	919	7.961
	C-4	25.183	3.698	2.397	31.196
	C-3	14.915	2.804	1.632	19.351
	C-1	31.719	3.728	2.744	37.691
		86.787	11.750	8.947	107.708
Mine 2	A-2	16.784	2.385	1.829	20.998
	B-3	14.838	1.338	1.609	18.181
	B-4	27.447	3.858	3.808	35.113
	B1-A	23.241	3.389	3.290	30.721
	C-4	18.280	3.887	3.457	25.624
	C-1	12.577	3.795	2.356	18.728
		131.589	18.809	16.459	166.860
Mine 3	A-2	48.277	7.164	8.080	63.451
	B-4	30.894	4.588	2.880	38.359
	B1-A	16.815	3.813	2.897	23.525
	C-4	19.979	3.981	1.329	25.289
	C-1	45.911	5.959	4.099	56.008
		161.876	25.184	19.284	195.705
Mine 4	A-1	31.114	4.277	3.027	38.417
	B1-B	23.196	3.488	1.580	28.263
	C-4	28.875	3.985	3.377	77.377
		113.284	15.729	9.273	137.857
TOTAL		493.540	69.877	60.010	617.319
Production Recap		85.53	11.93	8.13	105%

from
Longwall Continuous Miners in
Panels Room and Pillar Development

TABLE 3-2

RECOVERABLE COAL RESERVES BY MINE AND BY LESSOR

(Millions of Tons)

STATE OF UTAH COAL LEASE LANDS

Seam	Mine 1	Mine 2	Mine 3	Mine 4	Total
A-2	-	-	15.283	-	15.283
A-1	-	-	-	-	-
R-6	-	-	-	-	-
R-4	-	3.242	11.602	-	14.843
R1-B	-	-	-	1.782	1.782
R1-A	-	5.572	4.951	-	10.523
C-8	-	-	5.976	10.528	16.504
C-6	2.944	4.628	-	-	7.572
C-3	-	.310	14.112	-	14.422
C-1	5.914	-	-	-	5.914
TOTAL STATE	8.858	13.752	51.924	12.310	86.843

FEDERAL PREFERENCE RIGHT LEASE APPLICATION LANDS

Seam	Mine 1	Mine 2	Mine 3	Mine 4	Total
A-2	-	20.970	44.168	-	65.138
A-1	-	-	-	38.417	38.417
R-6	-	18.181	-	-	18.181
R-4	12.254	31.880	26.657	-	70.791
R1-B	-	-	-	26.381	26.381
R1-A	9.061	25.159	16.577	-	50.797
C-8	-	-	18.573	60.749	79.322
C-6	28.252	28.996	-	-	57.248
C-3	17.561	27.922	41.896	-	87.379
C-1	31.718	-	-	-	31.718
TOTAL FEDERAL	98.846	153.108	147.871	125.547	525.372
COMBINED	107.704	166.850	199.795	137.857	612.216

TABLE 3-1

RECOVERABLE COAL RESERVES BY MINE AND BY LESSOR

(Millions of Tons)

STATE OF UTAH COAL LEASE LANDS

Seam	Mine 1	Mine 2	Mine 3	Mine 4	Total
A-2	-	-	15.283	-	15.283
A-1	-	-	-	-	-
R-3	-	-	-	-	-
R-4	-	3.242	11.602	-	14.843
R1-B	-	-	-	1.782	1.782
R1-A	-	5.272	4.951	-	10.223
C-3	-	-	5.976	10.328	16.304
C-2	2.984	4.238	-	-	7.222
C-1	-	.370	14.172	-	14.422
C-1	3.918	-	-	-	3.918
TOTAL STATE	6.858	13.152	51.924	12.110	84.843

FEDERAL PREFERENCE RIGHT LEASE APPLICATION LANDS

Seam	Mine 1	Mine 2	Mine 3	Mine 4	Total
A-2	-	29.970	44.183	-	45.153
A-1	-	-	-	34.417	34.417
R-3	-	16.187	-	-	16.181
R-4	12.259	31.580	16.457	-	70.291
R1-B	-	-	-	24.384	24.384
R1-A	6.041	25.139	15.577	-	50.757
C-3	-	-	16.573	60.748	77.322
C-2	28.227	28.556	-	-	57.284
C-1	17.361	22.822	41.886	-	82.273
C-1	31.218	-	-	-	31.218
TOTAL FEDERAL	94.946	153.108	142.871	125.547	525.372
COMBINED	101.704	166.260	194.795	137.657	612.216

The mine layout prescribed for the Garfield Deposit does not favor early production of either State or Federal coal lands, rather a middle approach is assumed wherein both lessors would host mining activity at relatively equal levels throughout the productive life of the mines. No attempt to locate surface facilities on either type of lands was made, because it is assumed that the site use and environmental controls required would be equivalent, as would mining practices, coal recovery factors, etc.

The coal recovery factors in Table 3-3 were calculated for each seam, using the Total Reserves data presented in Section 2 and the Recoverable Coal Reserves presented in Table 3-1. Again, note that only 10 feet of coal is mined even in the seams which run over 20 feet in thickness, and no fringe or margin coal is extracted, both factors which will significantly reduce the recovery ratio. It is likely that these two sources of lost or nonextractable coal will actually be mined, but this preliminary mine plan does not include these mining activities. Thus, a conservative estimate of recoverable coal reserves was prepared, leading to a recovery factor which appears to be less than optimal.

Production Requirements

Production levels from the property are dictated by the "diligent development" criteria established by the Department of Interior as part of their leasing program. The specific dictates of this requirement are:

- that initial production be accomplished within ten years of lease issuance
- that 2.5% of the recoverable reserves be mined within that first ten years
- and that the remaining 97.5% be mined before the expiration of the 40-year lease.

TABLE 3-3

ESTIMATED COAL RECOVERY FACTORS

(Millions of Tons)

Seam	Total Reserves	Planned Recovery by Mine 1	2	3	4	Total	% Recovery
(A-1B) A-2	166.978	-	20.97	59.451	-	80.421	48.2
(A1-A) A-1	86.924	-	-	-	38.417	38.417	44.2
R-6	48.419	-	18.181	-	-	18.181	37.5
R-4	145.289	12.254	35.122	38.259	-	85.635	58.9
R1-B	56.048	-	-	-	28.163	28.163	50.2
R1-A	121.805	9.061	30.731	21.528	-	61.320	50.3
C-8	218.017	-	-	24.549	71.277	95.826	44.0
C-6	239.655	31.196	33.624	-	-	64.820	27.0
C-3	287.608	17.561	28.232	56.008	-	101.801	35.4
C-1	128.594	37.632	-	-	-	37.632	29.3
	1,499.337	107.704 MM tons from 5 seams	166.86 MM tons from 6 seams	199.759 MM tons from 5 seams	137.857 MM tons from 3 seams	612.216 MM tons from 10 seams	40.8%

To attain complete extraction of the 525 million recoverable tons of Federal coal (Table 3-2) according to the diligent development criteria will require that 13 million tons be mined during the first ten years development with 512 million mined in the final 30 years. At full production, 17.1 million tons per year must be mined. To compensate for the inevitable production slowdown as reserves are depleted, an 18 million ton per year rate has been selected for design purposes requiring 5 million TPY from three of the mines with 3 million TPY from the fourth. Table 3-4 describes the calculation of production capacity required by the diligent development regulations.

Table 3-4 also describes the previously mentioned Logical Mining Units. Based on coal tonnage found within these preliminary mining blocks, a nominal mine capacity was selected for each block. The mine blocks, or LMU's, are definitely not the only layout possible, and are probably not optimized, but represent a logical and workable layout. It should be noted that the mine areas are not perfectly separated by an arbitrary line along a section line, but are really overlapping because of the geometry of certain seams. A few of the coal seams are accessible in a coal seam level from only one mine, so it is projected to be mined from that mine rather than require ramp development through rock to gain access to a small coal seam area. Figure 3-1 illustrates the general layout of the four mines proposed for the Garfield Project.

Table 3-4 further lists the approximate production machinery complement projected for each mine, explaining how the various reserve blocks relate to production capacity and to production equipment requirements.

Table 3-5 described the geometry of each mine area, listing the thickness ranges of each coal seam and each overburden or interburden interval.

TABLE 3-4

DETERMINATION OF MINE PRODUCTION CAPACITIES AND REQUIRED MINING UNITS

Total Reserve	=	612 million tons
less State lands	=	87 million tons
Reserves on Federal lands	=	525 million tons

Diligent Development	2.5% during first ten years	= 13	million tons
	remainder	= 512	million tons
	Annual Production, 30 years	= 17.1	million tons

LOGICAL MINING UNITS

By topography, geologic continuity, access and haulage optimization

3 large reserve blocks with 160 to 200 million tons each could support 5 MMTPY mines

1 smaller reserve block with 100 million tons could support a 3 MMTPY mine with mine life of 33 to 36 years

Usual operating complement consists of:

1 long wall section: 1500 TPS x 2 shifts x 230 days/year = 690,000 tons/yr
2 continuous miners: 2 x 330 TPS x 2 shifts x 230 days/year = 303,600 tons/yr

993,600 tons/yr
or approximately 1 MMTPY

larger mines: 5 MMTPY capacity - 1 MMTPY per 3 unit complement = 5 complements, or 5 longwalls and 10 continuous miners

smaller mine: 3 MMTPY - 1 MMTPY = 3 complements, or 3 longwalls and 6 continuous miners

3 larger mines	= 3(5LW + 10CM)	= 15LW + 30CM
1 smaller mine	= 1(3LW + 6CM)	= 3LW + 6CM
4 mine total production units		= 18LW + 36CM

TABLE 3-5

SEAM GEOMETRY FOR EACH MINE

Mine #1

	Depth (Ft.)	Coal (Ft.) Thickness
Overburden	420-620	
R-4 Seam		5-15
Interburden	35-55	
R1-A Seam		7-10
Interburden	80-100	
C-6 Seam		5-15
Interburden	49-80	
C-3 Seam		8-25
Interburden	40-90	
C-1 Seam		15-20

Mine #2

	Depth (Ft.)	Coal (Ft.) Thickness
Overburden	300-920	
A1B Seam		5-11
Interburden	90-120	
R-6 Seam		5-15
Interburden	50-70	
R-4 Seam		5-12
Interburden	50-70	
R1-A Seam		5-11
Interburden	90-130	
C-6 Seam		5-12
Interburden	20-70	
C-3 Seam		5-16

Mine #3

	Depth (Ft.)	Coal (Ft.) Thickness
Overburden	370-700	
A1B Seam		5-7
Interburden	80-140	
R-4 Seam		5-10
Interburden	30-70	
R1-A Seam		5-11
Interburden	60-160	
C-8 Seam		5-12
Interburden	80-130	
C-3 Seam		10-21

Mine #4

	Depth (Ft.)	Coal (Ft.) Thickness
Overburden	160-590	
A1A Seam		5-13
Interburden	150-200	
R1-B Seam		7-10
Interburden	70-110	
C-8 Seam		5-21

TABLE 3-3

SEAM GEOMETRY FOR EACH MINE

Mine #1			Mine #2	
	Depth (Ft.)	Coal (Ft.) Thickness		Depth (Ft.)	Coal (Ft.) Thickness
Overburden	420-820		Overburden	300-320	
R-6 Seam		5-15	A1B Seam		5-11
Interburden	35-55		Interburden	90-120	
R1-A Seam		7-10	R-6 Seam		5-15
Interburden	80-100		Interburden	50-70	
C-6 Seam		5-15	R-4 Seam		5-12
Interburden	40-80		Interburden	50-70	
C-3 Seam		8-25	R1-A Seam		5-11
Interburden	10-30		Interburden	90-130	
C-1 Seam		15-30	C-6 Seam		5-12
			Interburden	20-70	
			C-3 Seam		5-15

Mine #3			Mine #4	
	Depth (Ft.)	Coal (Ft.) Thickness		Depth (Ft.)	Coal (Ft.) Thickness
Overburden	170-760		Overburden	150-590	
A1B Seam		5-7	A1A Seam		5-13
Interburden	80-140		Interburden	150-200	
R-4 Seam		5-10	R1-B Seam		7-10
Interburden	50-70		Interburden	70-110	
R1-A Seam		5-11	C-8 Seam		5-21
Interburden	60-150				
C-8 Seam		5-12			
Interburden	80-130				
C-3 Seam		10-11			

Figure 3-2 illustrates this geometry by featuring the coal seams to be mined. The seams are connected by the main slope which provides access and coal haulage for each seam level.

Figure 3-3 further illustrates the specific seam-by-seam development of a typical mine in the Garfield Project. Each overlay shows the mineable seam occurrence, and the planned mining area and pattern, while the entire assembly of overlays shows how the multiple-seam mine layout would be superimposed.

3.15 Organization of the Mining Complex

It is envisioned that each of the four mines would be managed by separate supervisory-engineering-administrative-maintenance staffs. The size of these individual mines requires that a fully capable authority be established at each mine, with adequate space and equipment to operate independently with only a few exceptions.

A central shop/office building is projected to house the management team and common administrative functions for the whole project. Supervisory, administrative, and engineering policies and priorities, special projects, regulatory liaison, and corporate liaison would be the role of the central office staff. Major maintenance such as overhauls and rebuilds should be handled at a well-equipped shop which would operate much as an outside vendor's shop with internal cost and priority controls.

The detailed organization is not fixed in this report because the costs of management infrastructure will vary insignificantly with the organizational structure. In Section 3.4 a listing of central and mine staffs is presented as a part of the estimation of labor costs for the project.

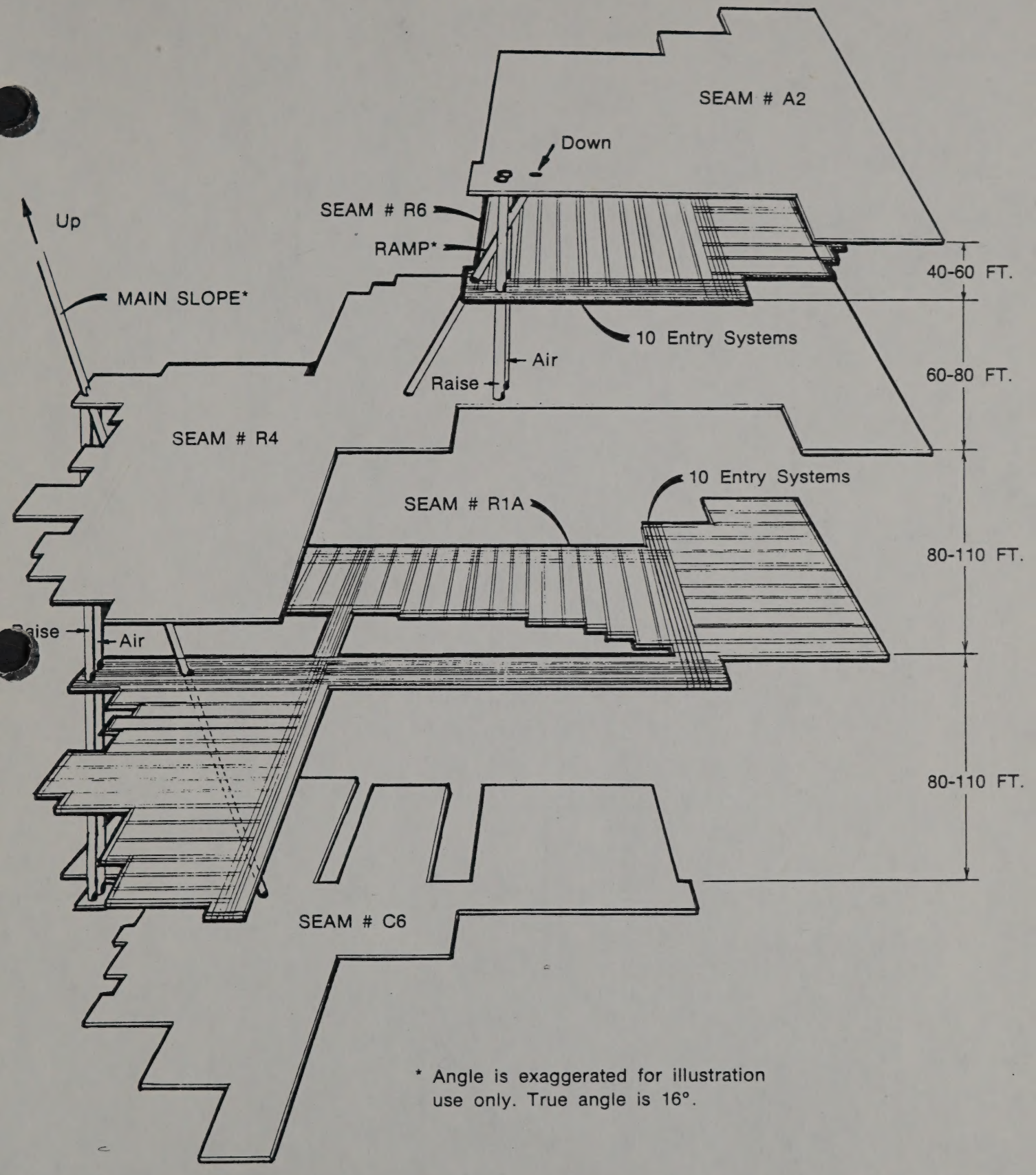
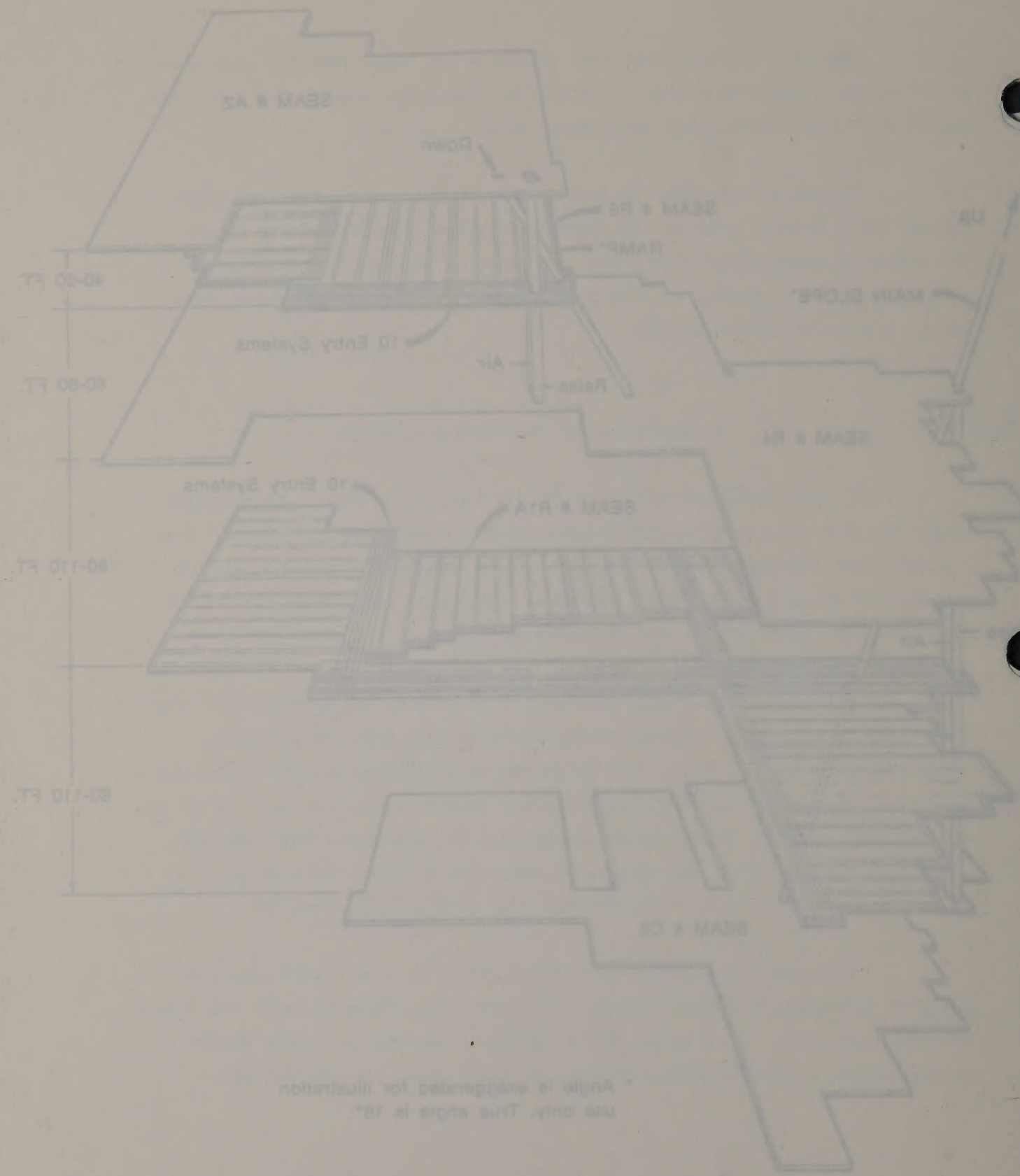


FIGURE 3-2: Perspective of Multiple-Seam Mine With Slope (Mine No. 2)

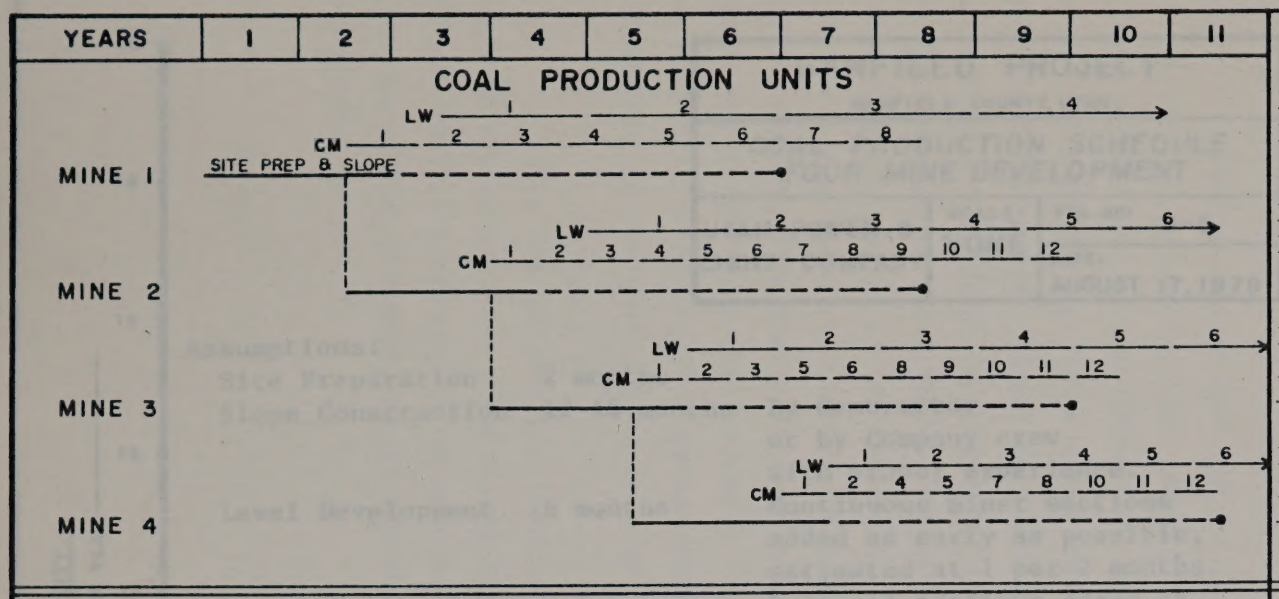


GARFIELD PROJECT
UTAH POWER & LIGHT COMPANY
DATE AUGUST 22 1973

Development Schedule

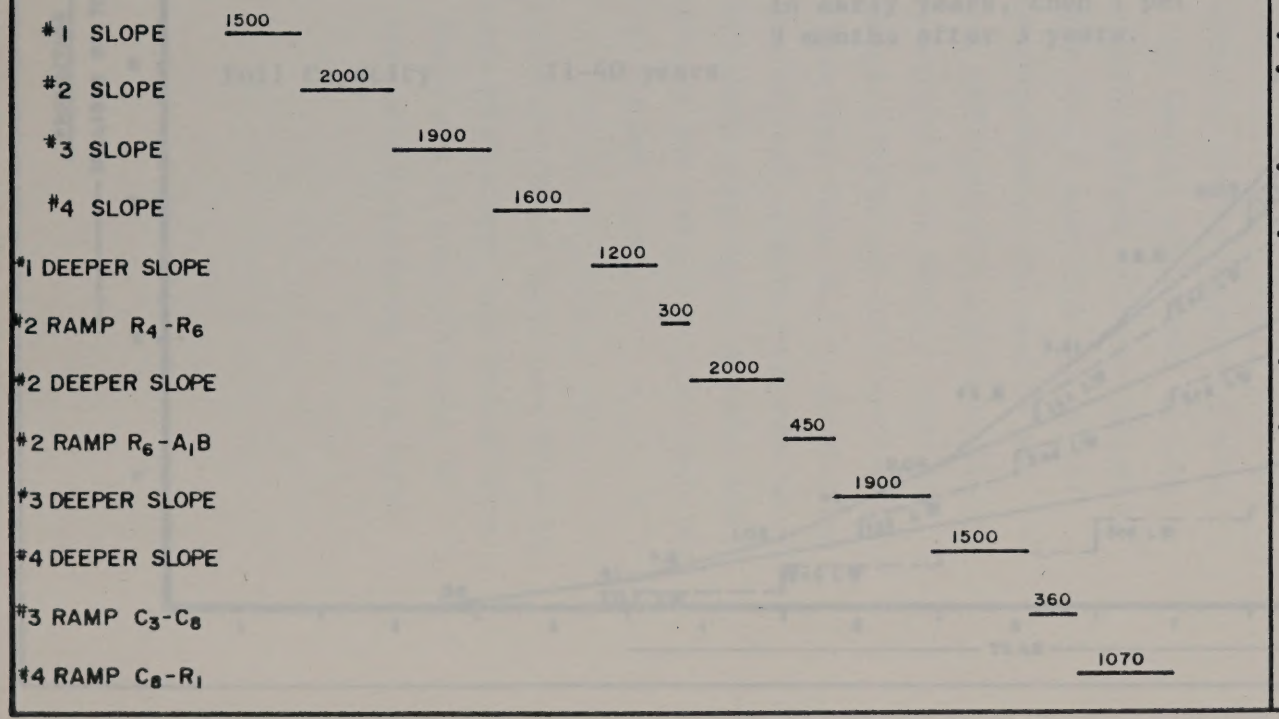
Figure 3-4 shows a probable development schedule resulting in full production of about 18 million tons per year during the eleventh year of the project. Total production to that point will be 58 million tons, well above the 13 million tons required under the diligent development criteria, because a slower startup would lead to an even higher long-term mine production level. A more detailed summary of production unit additions, rock work and surface development schedules is presented in Figure 3-5. The developmental sequence of the mine property will generally be from the southern end to the northern end in a stepwise fashion beginning with Mine No. 1 in the south and ending with Mine No. 4 at the head of Alvey Wash a few miles from Escalante. This sequence is required by the routing of the railroad which comes into the property from the south. Progressive mine development from the north to the south would necessitate complete construction of the overland belt which would not be used to capacity for several years after completion, or extension of the railroad to the north end of the property. These alternatives would be expensive both in construction capital necessary and in loss of coal left in place to control subsidence.

Development will progress such that the access road from Escalante is upgraded and paved early enough to haul equipment in from construction of the loadout facility. Other construction operations such as slope development and the office building will be able to progress concurrently with road upgrading. The road is now passable and will remain so during reconstruction. Slope construction at Mine No. 1 will commence immediately on arrival of the equipment on site and will progress concurrently with construction of the surface crushing and coal handling facilities, the mine office and shop and the railroad loadout. An overland belt will be built from the portal to the railroad loadout. This belt and Mine No. 1 are unique in the complex in not being connected with the other mines. Therefore, coal loadout can always continue from Mine No. 1 when the overland belt from the other mines



Assumptions:

- Site prep and slope/shaft/bottom construction = 2 years.
- Production buildup = 4 years to full capability.
- 10 year development period, full capacity in 11th year.



Assumptions:

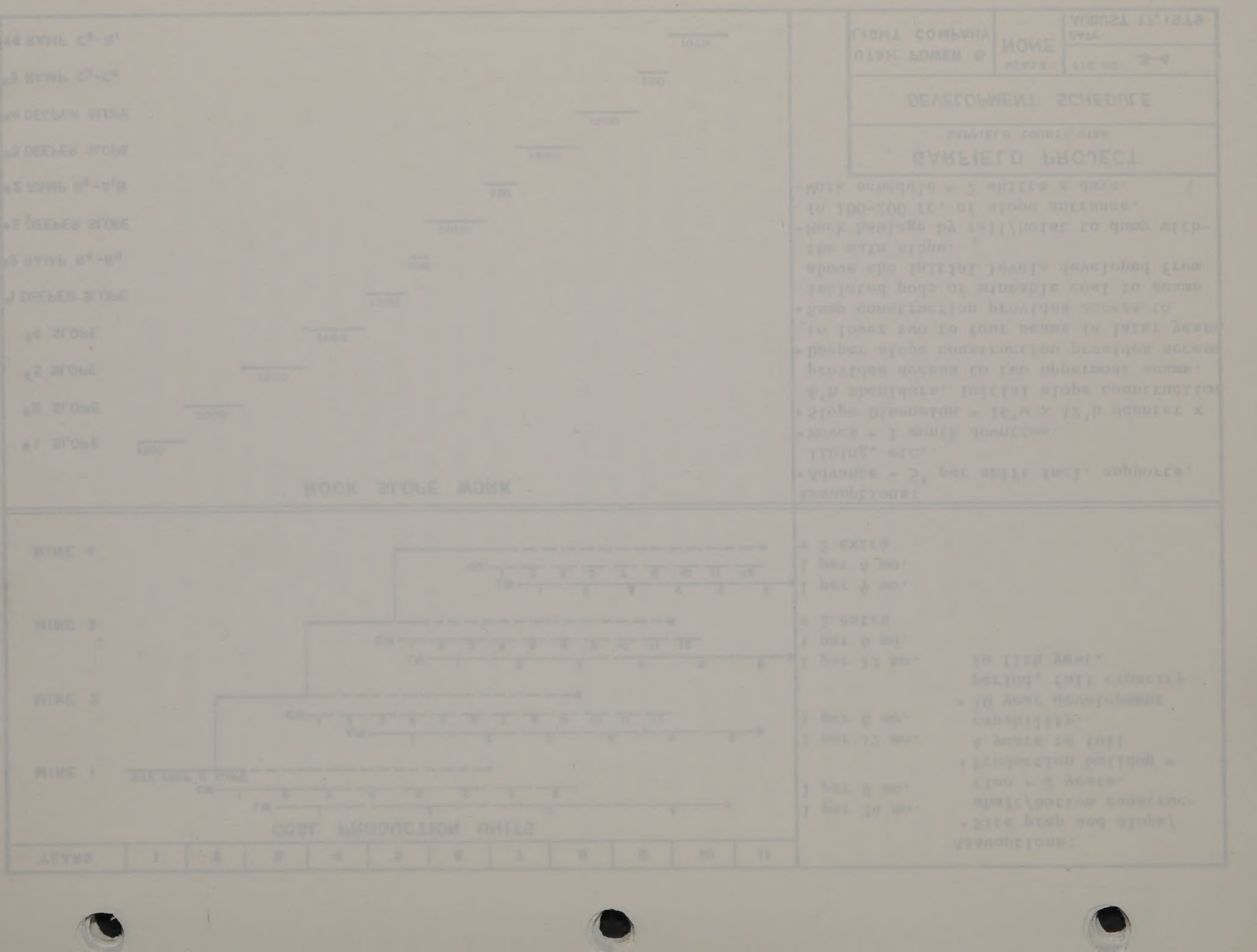
- Advance = 5' per shift incl. supports, lining, etc..
- Moves = 1 month downtime.
- Slope Dimension = 16'w x 12'h @center x 6'h shoulders. Initial slope construction provides access to two uppermost seams.
- Deeper slope construction provides access to lower two to four seams in later years.
- Ramp construction provides access to isolated pods of mineable coal in seams above the initial levels developed from the main slope.
- Muck haulage by rail/hoist to dump within 100-200 ft. of slope entrance.
- Work schedule = 2 shifts x days.

GARFIELD PROJECT

GARFIELD COUNTY, UTAH

DEVELOPMENT SCHEDULE

UTAH POWER & LIGHT COMPANY	SCALE: NONE	FIG. NO: 3-4
		DATE: AUGUST 17, 1979



GARFIELD PROJECT

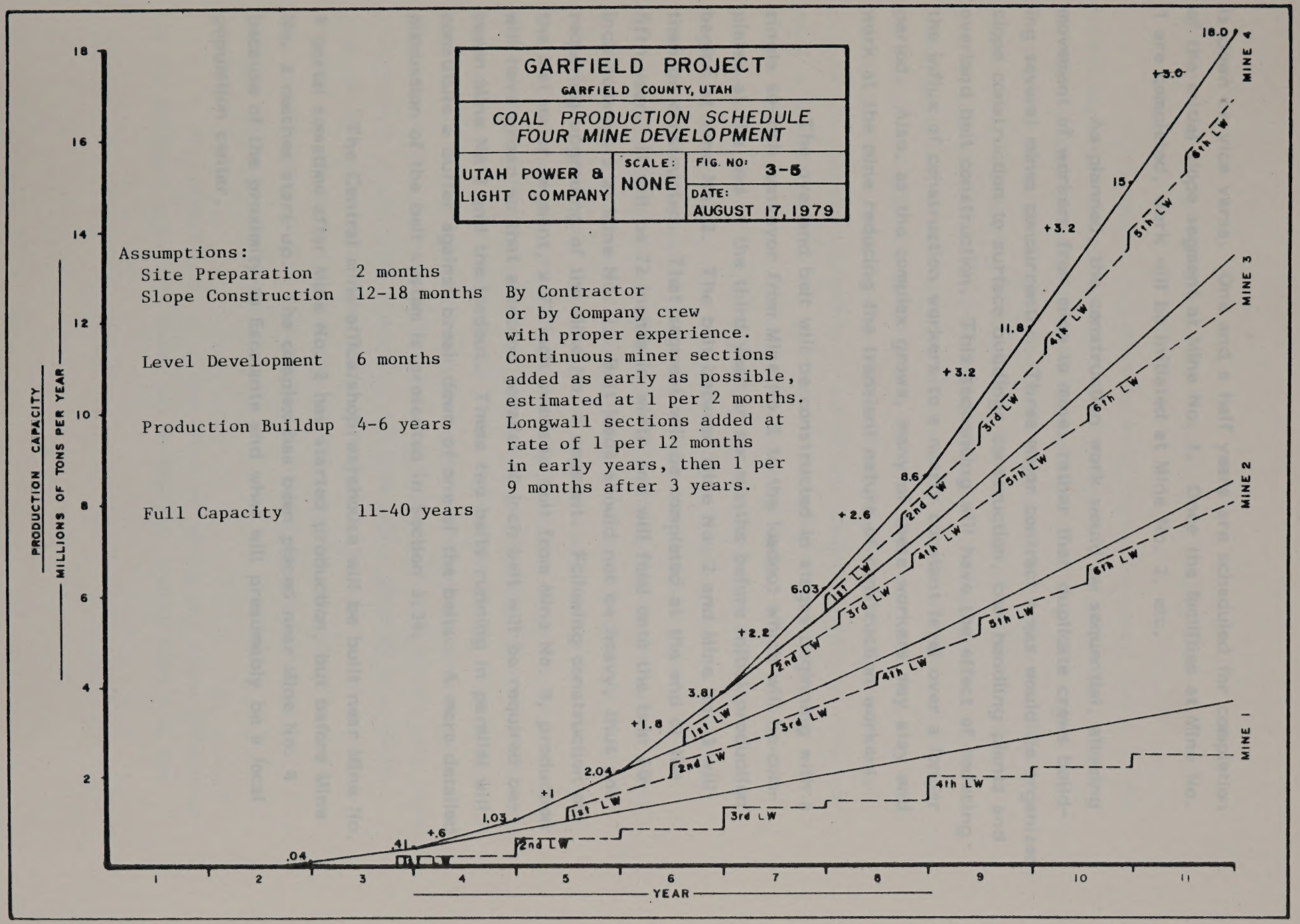
GARFIELD COUNTY, UTAH

COAL PRODUCTION SCHEDULE
FOUR MINE DEVELOPMENT

UTAH POWER & LIGHT COMPANY	SCALE: NONE	FIG. NO: 3-5
		DATE: AUGUST 17, 1979

Assumptions:

Site Preparation	2 months	
Slope Construction	12-18 months	By Contractor or by Company crew with proper experience.
Level Development	6 months	Continuous miner sections added as early as possible, estimated at 1 per 2 months.
Production Buildup	4-6 years	Longwall sections added at rate of 1 per 12 months in early years, then 1 per 9 months after 3 years.
Full Capacity	11-40 years	



is down or vice versa. One and a half years are scheduled for completion of the initial slope segment at Mine No. 1. Once the facilities at Mine No. 1 are completed, work will be initiated at Mine No. 2, etc.

As planned, the construction work would be sequential, allowing movement of workers from mine to mine, rather than duplicate crews building several mines concurrently. Three major contract areas would be organized: slope construction to surface buildings construction, coal handling plants and overland belt construction. This sequencing will have the effect of reducing the influx of construction workers to a nearly constant level over a longer period. Also, as the complex grows, many of these workers may stay and work at the mine reducing the transient nature of construction workers.

The overland belt will be constructed in stages beginning with a single 60-inch conveyor from Mine No. 2 to the loadout which will be completed at the end of the third year or six months before initial production begins at Mine No. 2. The belt between Mine No. 2 and Mine No. 3 will then be constructed. That segment will be completed at the end of the fifth year. It will be 72 inches in width and will feed onto the twin 60-inch conveyors at Mine No. 2. Belt loads should not be heavy, thus not requiring upgrading of the Mine No. 2 segment. Following construction of the last major segment, a 72 inch overland belt from Mine No. 4, production will have grown so that a second parallel 60-inch belt will be required between Mine No 2 and the loadout. These two belts running in parallel will constitute a buffer against break down of one of the belts. A more detailed discussion of the belt system is presented in Section 3.34.

The Central mine office/shop/warehouse will be built near Mine No. 4 portal sometime after Mine No. 2 has started production, but before Mine No. 3 reaches start-up. The complex has been placed near Mine No. 4 because of the proximity to Escalante and what will presumably be a local population center.

3.17 Environmental and Regulatory Considerations

The Southern Utah Environmental Impact Statement issued in May 1979 deals with the development of four southern Utah coal fields proposed for Federal approval and off-lease but related activities. The region covered by this EIS includes parts of five Utah counties: Garfield, Iron, Kane, Washington, and Wayne. A cumulative development of 12 million tons per year of coal production by 1990 and a proposed 2,500 megawatt capacity power generator is the source of the impacts. Southern Utah coal fields covered in the EIS include the Henry Mountains, Kaiparowits Plateau, Alton and Kolob Fields. Projections of 12 MMTPY production is dependent upon approval by Federal authorities of mining and reclamation plans. Development is planned to occur in eight mines belonging to three different companies (El Paso, Resources, and Utah International). Proposed actions of the Southern Utah EIS are the submission of the mining and reclamation plans proposing eight mines, each of which is a new operation developing an existing Federal lease. Three production scenarios are analyzed: a low production level of less than 1 MMTPY produced with no Federal action, a medium production level of about 30 MMTPY based on approval of all detailed mining proposals and preliminary transportation proposals, and a high production level of 46 MMTPY based on full approval of production, development of expanded transport, two new power generation project, and possible production from other Federal and State lands under lease. Predictions are carried to the year 1990 for each of the scenarios and analysis is made for total cumulative site-related impacts, and related off-site socioeconomic impacts, development of expanded transport, two new power generation projects, and possible production from other Federal and State lands under lease. Predictions are carried to the year 1990 for each of the scenarios and analysis is made for total cumulative site-related impacts, and related off-site socioeconomic impacts.

The impact analysis of the effect of mining and power generation on the Southern Utah Coal fields can be logically broken down into two

major areas. Impacts on the environment, physical and biological, and socioeconomic are the result of the direct effects of mining and transportation and the increased population associated with them. A preliminary survey of possible environmental effects show that they would result principally from removal and disturbance of land area preventing usage for the life of the mines and generating stations. An additional area of land would be affected by the actual coal removal through subsidence, if any, over mined seams and by transport systems. Some of the environmental effects to be expected are:

- loss of wildlife winter range
- interference with animal migration patterns
- loss of individual animals through hunting, fishing, and accidents
- hydrologic disturbance
- slight air quality degradation due to increased dust along transport corridors and in communities
- some unacceptable emissions from power plant stacks
- loss of vegetation

The socioeconomic and cultrual impacts revolve around the influx of new workers with new problems into the area.

It is proposed that all these impacts can be minimized by careful predevelopment planning and successful reclamation following the decline of actual production under this proposal. Careful adherence to the required agency regulations concerning mine development should reduce permanent damage to the lowest possible level. Cultural and socioeconomic disruptions can be cushioned by mutual willingness and good communication during the early planning stages.

Impacts to be expected in the specific mine areas are the same as those considered in the regional analysis but concentrated into a smaller area. Direct environmental effects arising from mining and transport will be:

- loss of acreage to wildlife for cover, forage, and nesting
- impediments to animal migration especially along the railroad right-of-way and overland conveyor belt
- some interference with surface water flow
- increased fugitive dust levels
- possible changes in ground water flow patterns

These impacts can be lessened by careful consideration of the consequences of engineering decisions and detailed premine planning. Socioeconomic impacts will occur throughout the counties concerned but will be most intense in those communities which are chosen for residence by the mine related personnel and which are associated with railroad construction utilization. Escalante, nearest town to the mine, will be the most likely to be disrupted. Many newcomers may choose to reside in this town and the community may change radically. Expected impacts are:

- increased requirement for community services
- increased recreational facilities and usage of existing facilities
- increased housing requirements
- increased need for necessities (stores)
- increased utility usage
- increased demand for financial services

All these problem areas can be treated successfully if considerable pre-development consideration is made for the needs of both the existing population and the immigrant population prior to their initial contact.

The Garfield Project will require a very detailed site-specific environmental impact statement devoted to the impacts to be reasonably expected by coal mining, railroad construction, and their related effects.

Table 3-6 is presented to give a broad scale evaluation of the areas to be investigated. During the course of environmental studies in

the area, new specific requirements may be discovered. These follow the essentials of the Southern Utah Regional EIS discussed above.

Table 3-7 is intended to indicate areas of permit or license concern and the relevant agencies which regulate those areas, but does not presume to list every specific permit required for the Project. Costs of obtaining permits will vary, so only a broad preliminary estimate is given. Lead time to acquire permits for mines have escalated in some cases to 8 to 10 years so no realistic schedule is given for this process. Several permit requirements may be dependent upon results of long-term environmental monitoring programs similar to those required in the detailed EIS.

TABLE 3-6
ANTICIPATED SCOPE OF
GARFIELD PROJECT EIS

Area of Concern/Specific Study	Estimated Cost (thousands of dollars)
Air	$350
present air quality baseline	
potential changes in suspended particulate - locally and area wide	
potential emission sources and amounts - locally and area wide	
air quality changes due to increased population	
potential impacts in air quality under various mining production scenarios	
Water	$650
baseline ground water quality and availability	
baseline surface water quality and quantity	
stream conditions, flow patterns, and aquatic biology	
potential sediment sources	
potential dissolved solid and gaseous sources	
stream diversions - potential impingement on channels and standing water bodies	
legal studies of water acquisition for mine use	
mine effluent impacts on present water quality and quantity	
Land	$350
soils mapping	
geology - especially seismic and potential toxic strata	
reclamation potential of disturbed areas	
potential subsidence effects	
potential erodibility of disturbed areas	
landslide - hazard potential	
paleontological resources	
archaeological and historical sites	
impact of non-recovery of resources	
alluvial valley and prime farmland survey and study	
Biological	$250
baseline survey of fauna - species inventory	
special sensitive species and habitat study	
vegetation typing and mapping	
ecological survey of potential habitat loss and effects on carrying capacity of area	
behavioral study, animal use patterns	
Socioeconomic	$500
present conditions of economics	
present conditions of services	
present conditions of transportation	
estimate effects of each phase of development	
impact upon existing population make-up	
impact on physical facilities - institutions	
investigate possibilities for wholly new community development	
investigate effects of growth on existing communities	
	$2,100

TABLE 3-7

REGULATORY REQUIREMENTS

Area of Concern/Specific Topic	Relevant Agency
Air Quality	
fugitive dust	Environmental Protection Agency
emission control	Utah Air Conservation Regulations
Water Quality	
dissolved solids	Federal Water Pollution Control Act
dissolved oxygen	Utah Bureau of Water Quality
effluent standards	
maximum pollution	
temperature	
waste disposal	
Land Use	
primary Federal land owner authorization for Federal land use - reclamation, restoration	Bureau of Land Management
cultural and historical surveys	Utah State Antiquities Act
dams, water wells	Utah State Engineer
highway crossings right-of-way	Utah Dept. of Transportation
mineral production, protection	Utah Oil, Gas, Mining, Commission
protection paleontological resources	BLM and State
utility and service requirements	County Commissioners
Cultural Resources	
cultural resources	Federal and State Regulations
survey-review by Utah	Utah State Antiquities Act
state historic preservation officer	BLM and State Cooperative Agreement
Fish and Wildlife	
coordination of protection for song and migratory birds	Department of the Interior U.S. Fish and Wildlife Service
Endangered Species	
survey for endangered species or their habitat	Department of the Interior
Railroads	
extension or construction of new lines	Interstate Commerce Commission
Rights-of-Way	Utah Dept. of Transportation

Mine Layout and Parameters

The mine plan for the Garfield Project is based on a set of mine layout maps, mine planning parameters, and recoverable coal reserve calculations prepared by Mr. John Peperakis, Mining Consultant, Salt Lake City, Utah.

The parameters are listed in Table 3-8 , the mine layouts are illustrated on seam mining maps in Appendix 3-1, and the recoverable coal reserves are presented in Table 3-1.

Mr. Peperakis, in a narrative accompanying the mine layout maps, urged caution in the use of the recoverable coal reserve figures. He used the term "extraction potential" because the calculations are based on interpretive seam isopach contours constructed from widely spaced drill holes. No attempt was made to consider rock split trends, width of transition zones, or 'wants' (channels) within the mineable seam areas.

The entire mine layout is based on the longwall mining system with only necessary entry development by room and pillar systems, because widespread use of room and pillar methods could well compromise the mineability of a substantial portion of the reserve.

On the seam mine layout maps, (Appendix 3-1) several shaded areas may be noted. These areas contain coal of workable thickness but interburden above or below of less than 30 feet. It is proposed that such areas be mined by advancing longwall or by single entry retreat longwall systems. These are higher cost methods, but allow more complete recovery in limited areas without serious impact on average mine costs. The reserve tonnage in this category is:

Mine 1	C-3 seam	9.0 MM tons
Mine 3	R-1B seam	6.0 MM tons
Mine 4	C-6 seam	29.0 MM tons
	Total	44.0 MM tons

This tonnage is not included in the recoverable coal reserve tabulations because of its doubtful mineability.

TABLE 3-8

MINE PLANNING PARAMETERS

Prepared by John Peperakis, Mining Consultant

Mining Height	10 foot maximum 4.5 to 5.0 foot minimum
Coal Density	1800 tons per acre foot 82.6 pounds per cubic foot 24.2 cubic feet per ton 1.32 specific gravity
Longwall Panels	500 ft. average face width 2000 to 5000 ft. range of panel length
Entries	Mains - 10 entries, grouped in 3-4-3 pattern, with groups and workings separated by four 200 ft. wide barrier pillars Entry width = 20 ft., centers = 60 ft. Small pillars are acceptable because of pressure arch supported by the four barrier pillars Submains - 5 entries, 20 ft. width, 80 ft. center, protected by 200 ft. wide barrier pillar on either side Panels - 2 entries, 20 ft. width, 50 ft. centers, these allow intentional squeezing following longwall extraction on either side so that pressure concentrations in pillar areas are reduced or eliminated in the subjacent seams. Isolation barriers are provided between groups of panels to facilitate sealing against fire and/or for pressure control.
Seam Connections	Airshafts and raises, 20 ft. diameter Coal transfer raises, 20 ft. diameter Mantrip elevator and airway, 20 ft. diameter
Coal Haulage	Belt conveyors used to the maximum Slope haulage to surface
Slopes	16 ft. width, 16 ft. height, rectangular bottom and semi-circular top, belt in top compartment, rail in lower
Mineable Coal Seam Selection	30 ft. minimum separation 5 ft. minimum thickness
Alternate Mining Methods (which may be employed, but which are not included in this plan)	Advancing longwall possible in areas of less than 30 ft. interburden Thick seam longwall and continuous miner operation in areas over 10 ft. thick Room-and-pillar mining without pillar recovery in areas beneath railroad spur loop Room-and-pillar mining with pillar recovery where possible in areas too small to justify longwall setup
Mining Conditions Comparison	Walsum Mine, Ruhr area, West Germany capacity over 20,000 metric tons per day depth of 1200 meters (3900 feet) Selby Project, England now under development projected tonnage of 10 MMTPY from one haulage opening tapping a complex multiple-seam deposit

Seam Access 3.21

Table 3-5 describes the seam/overburden/interburden thicknesses found in each mine; Table 3-9 details the projected lengths of slope/shaft/ramp required to gain access to the coal seams.

Slopes will be used for principal access to the multiple seam coal deposits. Figure 3-2 illustrates the three-dimensional geometry of a typical mine, highlighting the slope access into each seam. Figure 3-6 describes a typical slope configuration. Main slopes will be compartmentalized into an upper semicircular section and a lower rectangular section. The horizontal partition will be reinforced concrete anchored into the concrete walls and top. The upper semicircular section have an 8 foot radius and contain the slope haulage 60 inch flexsteel type belt traveling at 450 feet per minute with 35° idlers. This belt will be off-set from the center so that a walkway can exist for required maintenance access.

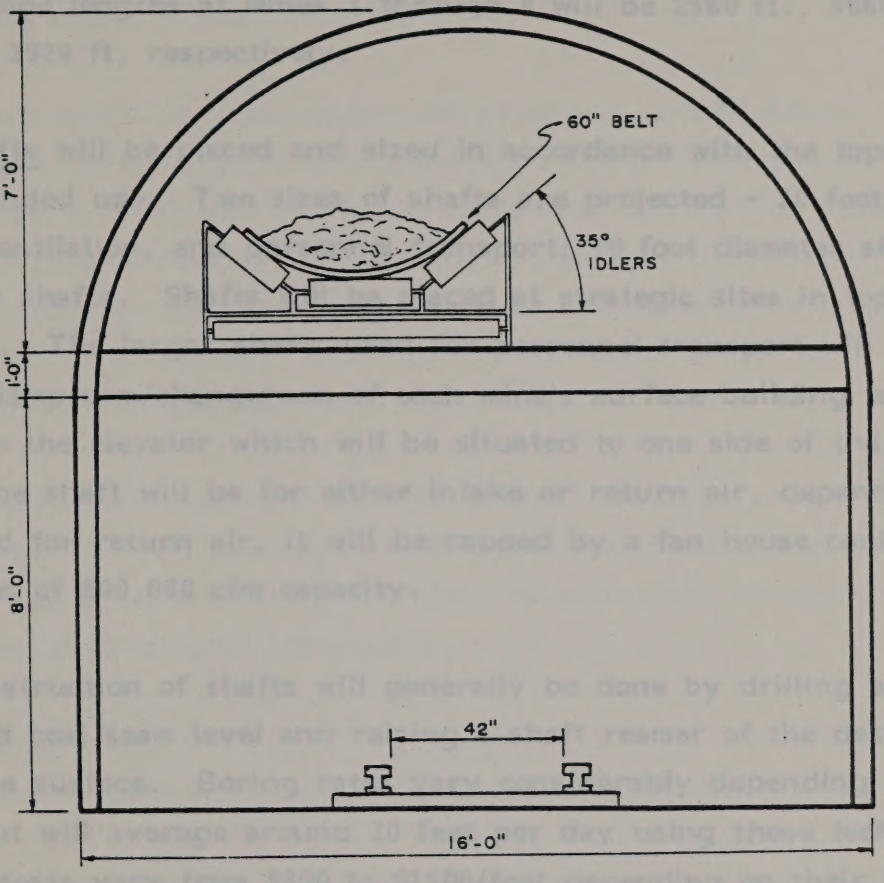
The lower rectangular portion of the slope, 9 feet by 16 feet, will be used for lowering supplies and equipment into the mine, pulling equipment out for repair, and as a fresh air intake. As designed, the slope will have 80 lb/yard, 42 inch gauge track on wooden and steel ties extending down the entire length of the slope with switches at each seam. Each slope will be equipped with a hoist of about 75 ton capacity. Commonly, the hoist will be used to lower loaded supply cars two or three at a time. Greater single loads are expected when handling the larger equipment pieces such as shearers and continuous miners. Each slope will have a grade of 16°. Coal will only be loaded onto the belt at the basal seam of each of the three main coal-bearing zones and an angled accelerator belt will be used.

Construction of the slopes is estimated to advance at a rate of approximately five feet per day at a cost of $2,500/foot which includes the

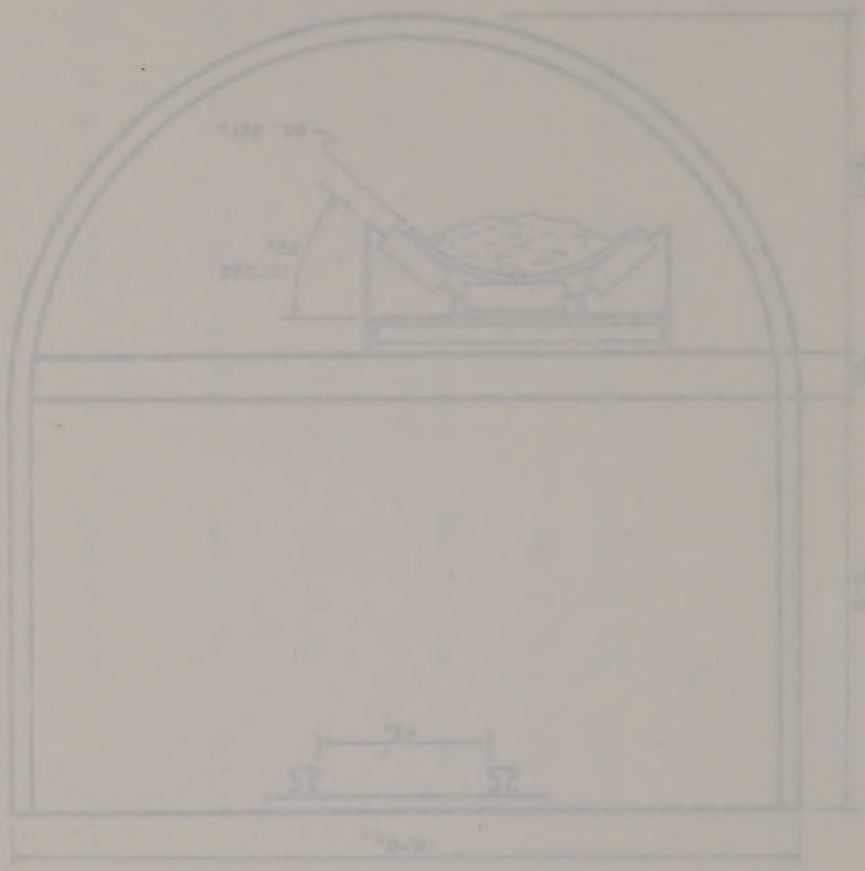
TABLE 3-9

SHAFT, SLOPE, AND RAMP REQUIREMENTS AND COSTS

		SHAFTS		SLOPES		RAMPS			
MINE	Deepest Seam/Depth	No. & Length	Cost (at 1200/ft.)	Length	Cost (at 2500/ft.)	Seams	Ramp Length	Cost (at 1000/ft.)	TOTALS
#1	C-1/705'	4 @ 705' = 2820'	$3.38 MM	2,560'	$6.40 MM	-	-	-	$ 9.78 MM
#2	C-3/1120'	1 @ 1040' 4 @ 1120' = 5520	$6.62 MM	4,060'	$10.15 MM	R6-A2 R4-R6	360' 260' 620'	$0.62 MM	$17.39 MM
#3	C-3/1050'	5 @ 1050' = 5250'	$6.30 MM	3,810'	$9.53 MM	C3-C8	520'	$0.52 MM	$16.35 MM
#4	C-8/805'	6 @ 805' = 4830'	$5.80 MM	2,920'	$7.30 MM	C8-R1B	730'	$0.73 MM	$13.83 MM
TOTALS		20 = 18,420'	$22.10 MM	4 = 13,350'	$33.38 MM	4 =	1870'	$1.87 MM	$57.35 MM



GARFIELD PROJECT GARFIELD COUNTY, UTAH		
TYPICAL SLOPE CONFIGURATION		
UTAH POWER & LIGHT COMPANY	SCALE: NONE	FIG. NO: 3-6
		DATE AUGUST 21, 1979



GARFIELD PROJECT

GARFIELD COUNTY, UTAH

TYPICAL SLOPE CONFIGURATION

UTAH POWER & LIGHT COMPANY

SCALE NONE

DATE AUGUST 21, 1979

FIG. NO. S-6

track and the concrete lining. Including a mobilization fee of about 10%, construction of these slopes can be done in two steps. The first phase would complete the slope to below the first two seams and the second would finish the slope. Slope lengths at mines 1 through 4 will be 2560 ft., 4060 ft., 3810 ft., and 2920 ft. respectively.

Shafts will be placed and sized in accordance with the topography and their intended use. Two sizes of shafts are projected - 20 foot diameter for primary ventilation, and personnel transport; 16 foot diameter shafts for ventilation shafts. Shafts will be placed at strategic sites in topographically low areas. The larger shafts used for personnel transport will be adjacent to the lamproom/changeroom of each mine's surface building to provide easy access to the elevator which will be situated to one side of the shaft. The rest of the shaft will be for either intake or return air, depending on need. If used for return air, it will be capped by a fan house containing an exhaust fan of 500,000 cfm capacity.

Construction of shafts will generally be done by drilling a hole down to a developed coal seam level and raising a shaft reamer of the desired size through to the surface. Boring rates vary considerably depending on the size of the hole but will average around 20 feet per day using those techniques. Raise boring costs vary from $800 to $1500/foot depending on their lined diameter.

Ramps will be required in Mine Nos. 2, 3, and 4 to provide access to isolated pods or splits of seams impossible to develop from the slope due to thinness of the coal or the geometry of the seam. For instance, seam splits will usually result in the continued mining of the lower bench of the seam. Once the two benches have separated by 30 feet, mining in both seams is considered possible, so a ramp is required to provide access to the upper bench. Functionally, the ramps will be used for supply and personnel access to the upper seam and as a ventilation way for the introduction of fresh air into the upper levels. The ramps will be equipped with

a hoist capable of raising two or three loaded supply cars up the 16° incline. The particular requirements for this hoist will not be as stringent as for the main slope hoist since maximum ramp length for servicing two such seam pods is 800 feet, and speed is much less important.

Construction of these ramps will involve a certain amount of rock work which must be done by crews similar to those performing slope construction. Costs are far less for these ramps and construction rates are far higher for several reasons: construction begins at the bottom of the ramp and proceeds up, mucking is much easier with gravity assist, concrete lining should not be necessary; stowing or rock waste can be done in nearby crosscuts, and without the belt, the size is smaller - 8ft. x 16ft.

Construction costs will be from $600 to $800/foot, while the advance rate will be about 10 feet per day. At the foot and head of the ramp there will be track connections which will act as a staging area for switching the supply cars for their proper destination.

Continuous Miner Development of Entries

Continuous miner units will be used solely for development work while longwalling techniques will be used for the actual mining. There are no plans to retreat mine any of the development chain pillars or adjacent barrier pillars primarily because of the danger involved in possibly cracking old panel seals. This mining scheme may be changed if the appropriate technology becomes available. There are several small areas beneath the railroad loadout facility in which only 50% extraction by a continuous miner will be allowed in order to reduce subsidence. Other surface facilities will not be susceptible to subsidence since they will either be placed over main entries or will be designed to withstand subsidence.

The process of mining with continuous miners involves a sequence

of moves of four basic pieces of equipment - a continuous miner, roof bolter and two shuttle cars. The continuous miner will usually take a 20-feet deep cut the width of the entry. While mining, the coal is dumped into shuttle cars, a batch haulage method carrying from 5 to 10 tons per load. These shuttle cars will then haul the coal no more than a few hundred feet to the belt head and dump it into a feeder breaker which acts as a surge bin and crusher while breaking the coal and feeding it to the section belt. After finishing a cut, the miner is moved to an adjacent cut, the ventilation is modified somewhat, then the roof bolter is moved into the first cut to bolt the immediate and primary roof. Table 3-10 lists and prices the equipment found in a typical continuous miner section.

In the process of developing the mine, initial bottom development must first be accomplished before entries can be developed. As a general case, bottom or level development will be done on the uppermost seam to be mined before work is initiated on any lower seam. This initial development process will require at least six months by a single unit during which time the slope bottom, bottom shop area, shaft bottom, mains, and any other development entries are outlined. Once outlined, three more continuous miner units may be introducted into the 10-entry mains. Additional units can be placed in service outlining longwall panels. Longwall units can be placed in service once enough panels are outlined to ensure that the longwall units can be constantly kept in coal. Removal of continuous miner units to the next subjacent level will follow achievement of stabilized production in the upper seam, but will generally only occur later in the mineable life of that upper seam.

During the mine development process, three different entry systems already alluded to will be employed with each having certain well-defined purposes. The mains will be made up of a 10-entry system while the submains will be 5-entry and the longwall panels will be outlined by 2-entry development.

TABLE 3-10

CONTINUOUS MINER SECTION COMPONENTS AND COSTS*

Quantity	Equipment	Costs
1	Continuous Miner	$ 413,000
2	Shuttle Cars	270,000
1	Roof Bolter	132,000
1	Feeder Breaker	81,000
1	Utility Scoop (Diesel)	120,000
1	Trickle Duster (Return)	6,000
1	Rock Duster (Face)	8,000
3	Supply Cars	66,000
1	Service Center	20,000
1	Mobile Shop	11,000
1	Mantrip (Battery)	80,000
1	1000/300 KVA Power Center	60,000
1	Welder	2,000
1	Pump	8,000
2	650' S.C. Cables	5,000
1	650' Miner Cable	7,000
1	650' Bolter Cable	3,000
1	500' Feeder Breaker Cable 2/0-4C	3,000
1	2000' Section Power Cable 8Kv 2/0-3C SHD	36,000
1	Lot Communication Equipment	2,000
1	Lot Face Lighting (Miner)	6,000
1	Lot Safety Equipment	5,000
1	First Aid Station	1,000
1	500 KVA Transformer	30,000
1	48" Belt Drive, 150 HP, Take-up and Tail	47,000
1	48" Belt 5000 ft.	90,000
1	48" Belt Structure - 2500 ft.	49,000
1	Fire Suppression System (Foam Maker, Sensor, etc.)	4,000
1	Lot of Water Pipe and/or Hose and Fittings	10,000
	Subtotal	$1,575,000
	Unidentified	158,000
	TOTAL	$1,733,000

*Includes components of water, supply, communications, electrical and coal haulage systems immediate to a continuous miner section.

TABLE 3-10

CONTINUOUS MINER SECTION COMPONENTS AND COSTS*

Quantity	Equipment	Costs
1	Continuous Miner	$ 413,000
2	Shuttle Cars	270,000
1	Roof Bolter	122,000
1	Feeder Breaker	81,000
1	Utility Scoop (Diesel)	120,000
1	Trickle Duster (Return)	5,000
1	Rock Duster (Face)	8,000
1	Supply Cars	66,000
1	Service Center	20,000
1	Mobile Shop	11,000
1	Mancrip (Battery)	80,000
1	1000/750 KVA Power Center	60,000
1	Welder	2,000
1	Pump	8,000
1	480' S.C. Cable	5,000
1	650' Miner Cable	7,000
1	650' Bolter Cable	3,000
1	100' Feeder Breaker Cable 210-6C	3,000
1	2000' Section Power Cable 8Kv 210-3C SHD	36,000
1	Lot Communication Equipment	2,000
1	Lot Face Lighting (Miner)	6,000
1	Lot Safety Equipment	5,000
1	First Aid Station	1,000
1	150 KVA Transformer	30,000
1	40" Belt Drive, 150 HP, Take-up and Tail	47,000
1	36" Belt 3000 ft.	90,000
1	60" Belt Structure - 3200 ft.	49,000
1	Fire Suppression System (Foam Maker, Sensor, etc.)	8,000
1	Lot of Water Pipe and/or Hose and Fittings	10,000
	Subtotal	$1,525,000
	Unidentified	158,000
	TOTAL	$1,733,000

*Includes components of water, supply, communications, electrical and coal haulage systems immediate to a continuous miner section.

Mains will be developed using the 10-entry system with three parallel sets of entries in a 3-4-3 pattern. A two hundred foot barrier pillar will be left between each set of three or four entries and between the ten entries and the workings on either side. It is desired to induce an arching effect such that overburden pressures will rest on the barrier pillars, thus relieving the entry pillars of large stresses. The process of actually driving the mains involves the parallel development of 3-entry, 4-entry and 3-entry portions of the 10-entry system by three continuous miner units. Figure 3-7 depicts these two portions of the 10-entry development system and the typical ventilation schemes and mining sequences that might be used. To determine a nominal advance rate for these entry types, tonnage per foot figures of 28 and 40 in 10 foot high coal can be calculated for the 3- and 4-entry portions respectively. Using a nominal 330 tons per shift results in advance rates of all entries of respectively 12 and 8 foot per shift or an average 100 shifts per 1000 feet.

Pillar dimensions are shown as being 40 ft. x 40 ft. in nominal dimensions on 60 ft. x 60 ft. entry and cross cut centerlines. Application of the arching theory is reflected in the small size of the pillars. It also reflects the roof control plan which calls for bolting the overlying shales and sandstones into a laminated beam primarily using four and five foot mechanical bolts. Resin type bolts will also be required in special circumstances.

Uses have been assigned to the entries as follows:

- one set of three entries will be for intake air
- the other set will be for return air
- the central four entries will be for belt haulage, rail supply haulage, sidetracks and rubber tire vehicles.

In designating uses for these entries allowance must be made for modification as needed. If it is found that a second parallel belt is needed, it may displace one of the other uses.

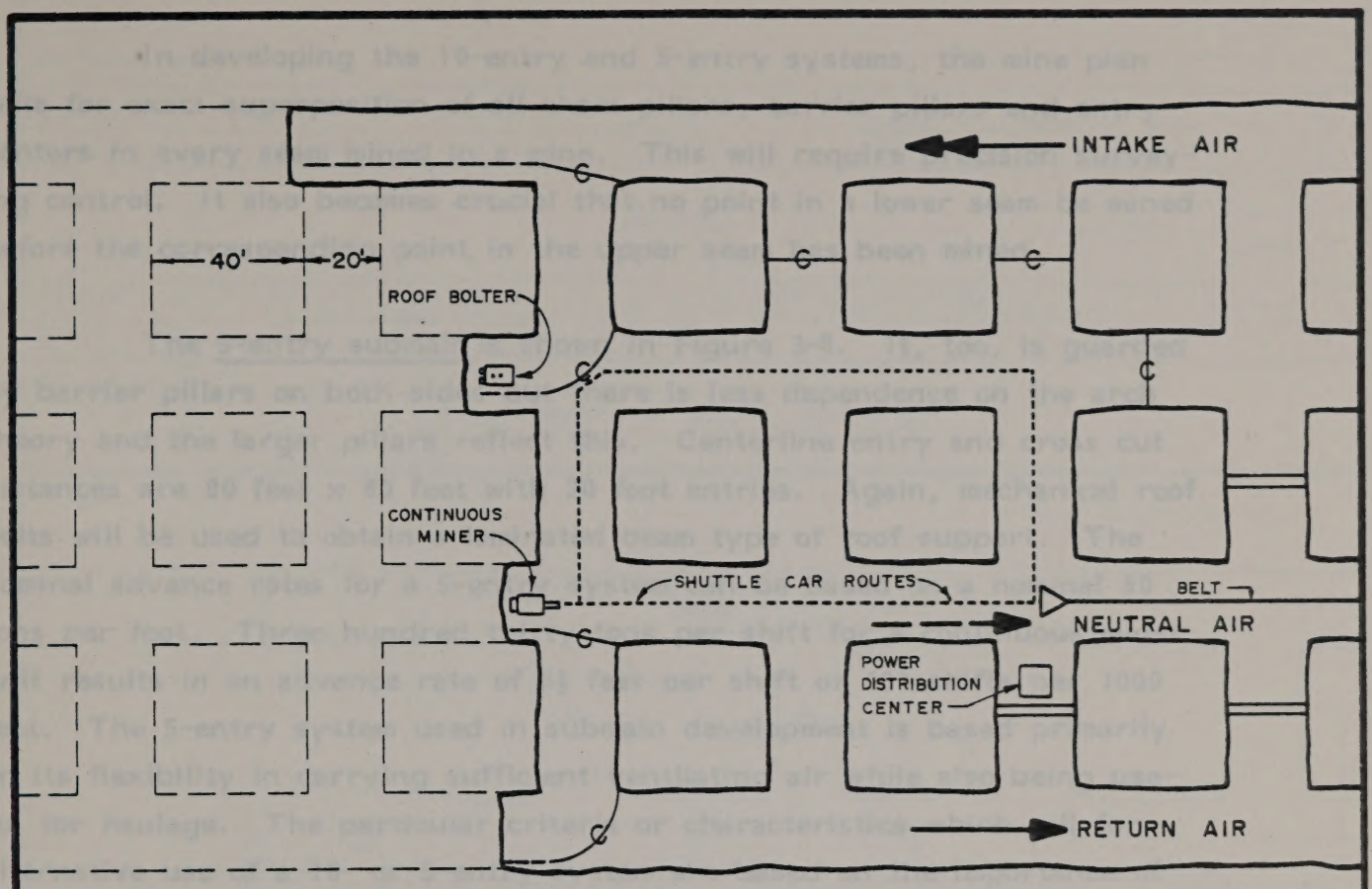
Mains will be developed using the 10-entry system with three parallel sets of entries in a 3-4-3 pattern. A two hundred foot barrier pillar will be left between each set of three or four entries and between the ten entries and the workings on either side. It is desired to induce an arching effect such that overburden pressures will rest on the barrier pillars, thus relieving the entry pillars of large stresses. The process of actually driving the mains involves the parallel development of 3-entry, 4-entry and 3-entry portions of the 10-entry system by three continuous miner units. Figure 3-7 depicts these two portions of the 10-entry development system and the typical ventilation schemes and mining sequences that might be used. To determine a nominal advance rate for these entry types, tonnage per foot figures of 22 and 30 in 10 foot high coal can be calculated for the 3- and 4-entry portions respectively. Using a nominal 320 tons per shift results in advance rates of all entries of respectively 12 and 8 foot per shift or an average 100 shifts per 1000 feet.

Pillar dimensions are shown as being 80 ft. x 80 ft. in nominal dimensions on 80 ft. x 80 ft. entry and cross cut centerlines. Application of the arching theory is reflected in the small size of the pillars. It also reflects the roof control plan which calls for bolting the overlying shales and sandstones into a laminated beam primarily using four and five foot mechanical bolts. Resin type bolt will also be required in special circumstances.

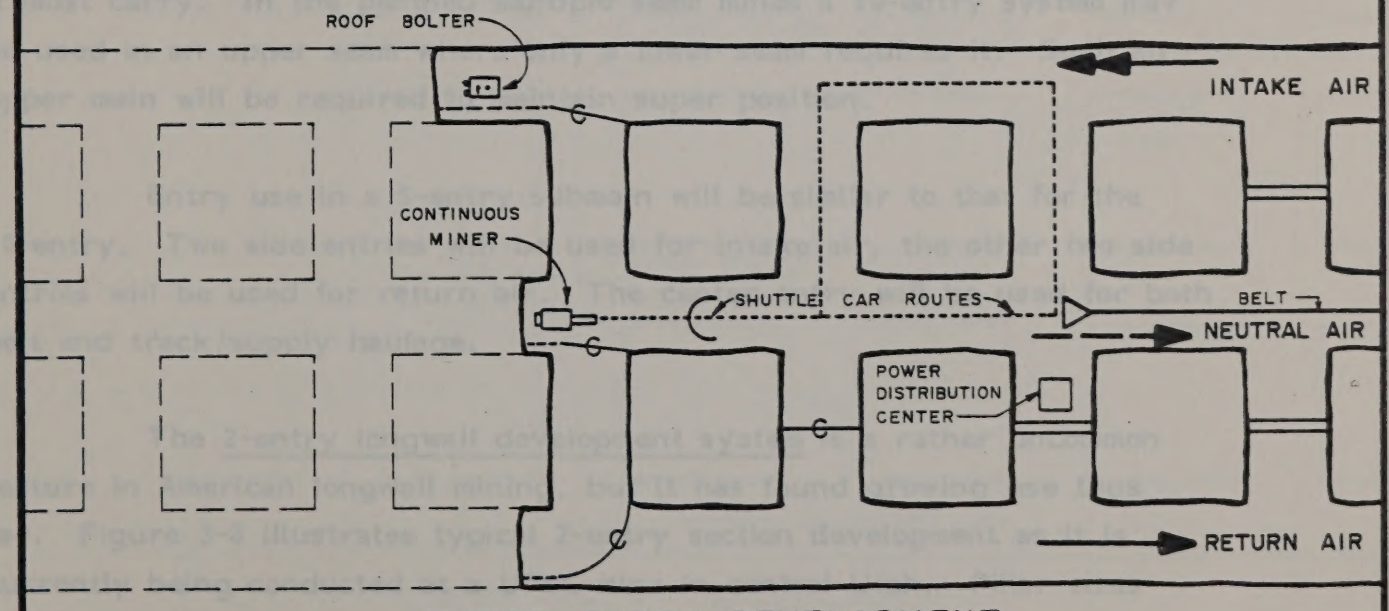
Uses have been assigned to the entries as follows:

- one set of three entries will be for intake air
- the other set will be for return air
- the central four entries will be for belt haulage, rail supply haulage, sidetracks and rubber tire vehicles.

In designating uses for these entries allowance must be made for modification as needed. If it is found that a second parallel belt is needed, it may displace one of the other uses.



FOUR ENTRY DEVELOPMENT



THREE ENTRY DEVELOPMENT

LEGEND

CHECK CURTAIN ----- LINE BRATTICE

TRAVEL CURTAIN ═══ STOPPING

GARFIELD PROJECT
GARFIELD COUNTY, UTAH

DEVELOPMENT OF MAIN ENTRIES BY CONTINUOUS MINERS

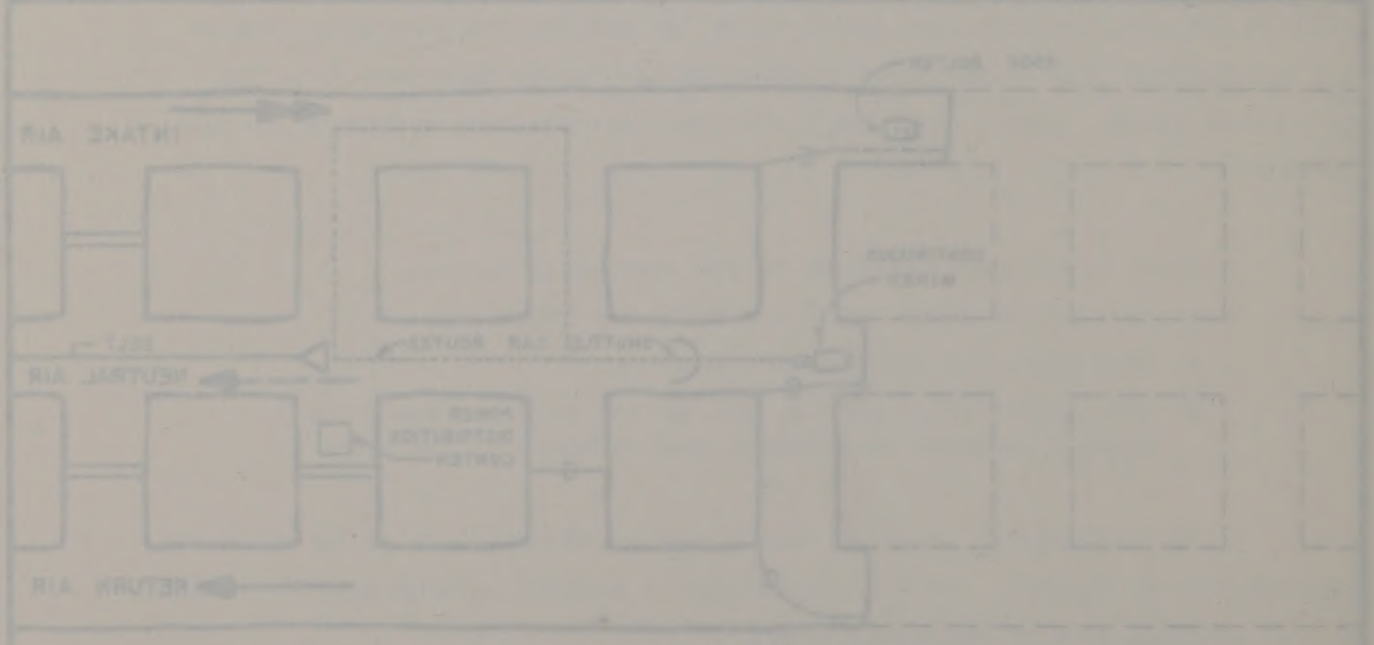
UTAH POWER & LIGHT COMPANY

SCALE: 1" = 50'

FIG. NO: 3-7

DATE: AUGUST 17, 1979

FOUR ENTRY DEVELOPMENT



THREE ENTRY DEVELOPMENT

GARFIELD PROJECT

DEVELOPMENT OF MAIN ENTRIES BY CONTINUOUS MINERS

In developing the 10-entry and 5-entry systems, the mine plan calls for exact superposition of all chain pillars, barrier pillars and entry centers in every seam mined in a mine. This will require precision surveying control. It also becomes crucial that no point in a lower seam be mined before the corresponding point in the upper seam has been mined.

The 5-entry submain is shown in Figure 3-8. It, too, is guarded by barrier pillars on both sides but there is less dependence on the arch theory and the larger pillars reflect this. Centerline entry and cross cut distances are 80 feet x 80 feet with 20 foot entries. Again, mechanical roof bolts will be used to obtain a laminated beam type of roof support. The nominal advance rates for a 5-entry system can be based on a nominal 50 tons per foot. Three hundred thirty tons per shift for a continuous miner unit results in an advance rate of $6\frac{1}{2}$ feet per shift or 154 shifts per 1000 feet. The 5-entry system used in submain development is based primarily on its flexibility in carrying sufficient ventilating air while also being useful for haulage. The particular criteria or characteristics which call for alternative use of a 10- or 5-entry system are based on the importance of the system to the mine in general, the length of time it must be available, the number of panels being served by it and the amount of ventilating air it must carry. In the planned multiple seam mines a 10-entry system may be used in an upper seam where only a lower seam requires it. Such an upper main will be required to maintain super position.

Entry use in a 5-entry submain will be similar to that for the 10-entry. Two side entries will be used for intake air, the other two side entries will be used for return air. The center entry will be used for both belt and track/supply haulage.

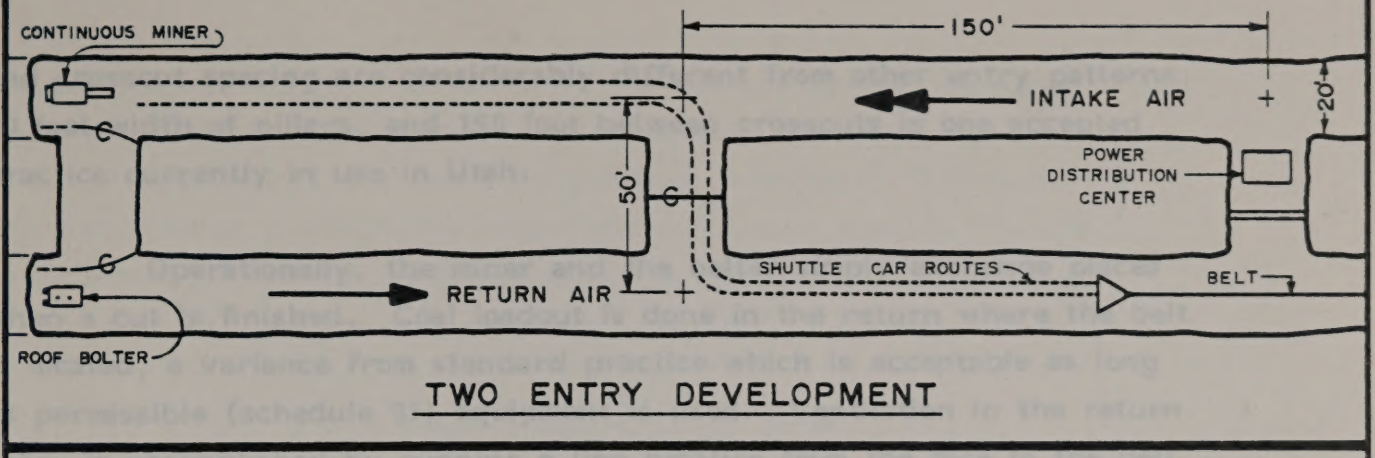
The 2-entry longwall development system is a rather uncommon feature in American longwall mining, but it has found growing use thus far. Figure 3-8 illustrates typical 2-entry section development as it is currently being conducted at a UP&L mine in central Utah. Pillar sizes

In developing the 10-entry and 5-entry systems, the mine plan calls for exact superposition of all chain pillars, barrier pillars and entry centers in every seam mined in a mine. This will require precision surveying control. It also becomes crucial that no point in a lower seam be mined before the corresponding point in the upper seam has been mined.

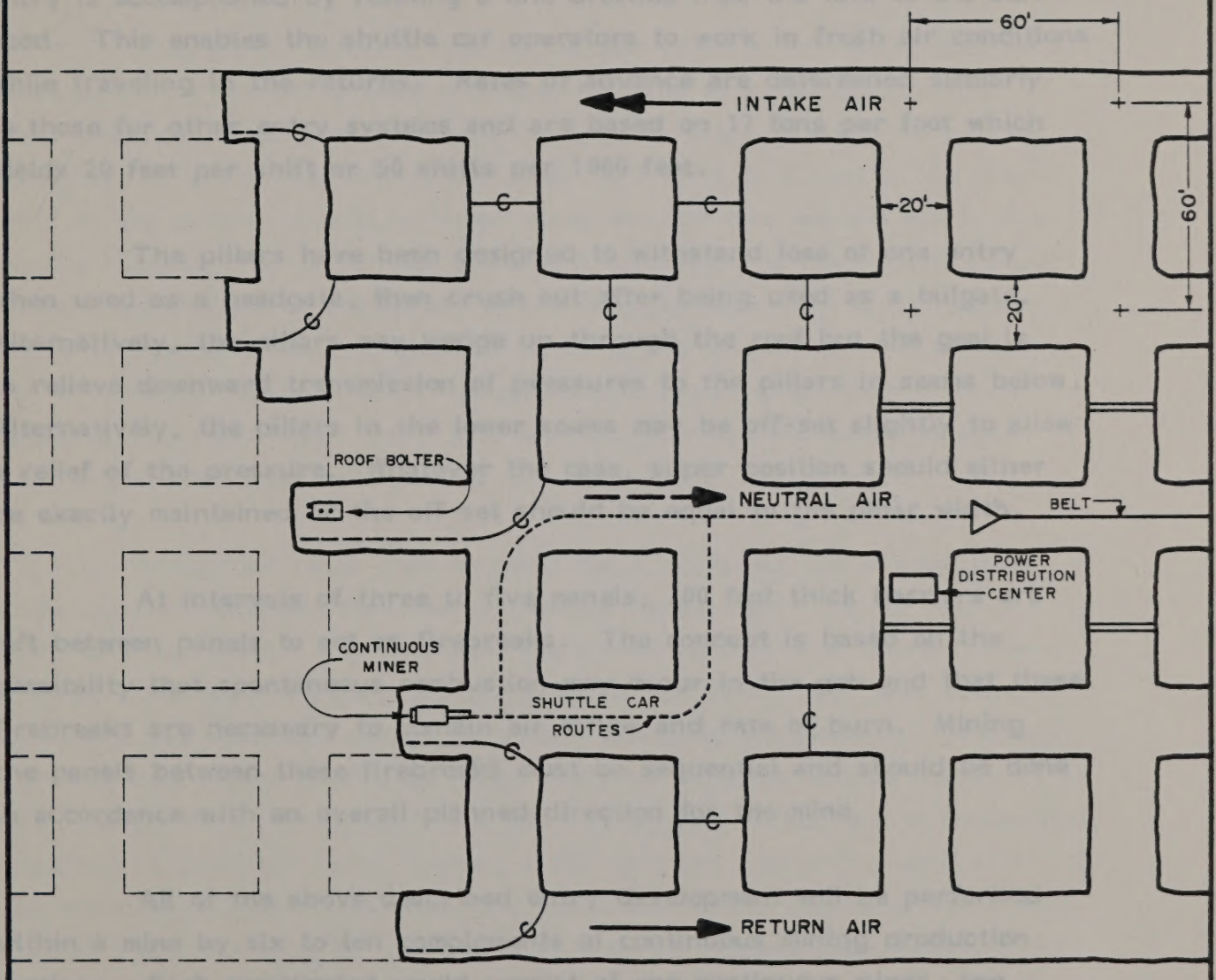
The 5-entry subplan is shown in Figure 3-6. It, too, is guarded by barrier pillars on both sides but there is less dependence on the arch theory and the larger pillars reflect this. Centerline entry and cross cut distances are 20 feet x 80 feet with 20 foot entries. Again, mechanical roof bolts will be used to obtain a laminated beam type of roof support. The nominal advance rate for a 5-entry system can be based on a nominal 50 tons per foot. Three hundred thirty tons per shift for a continuous miner unit results in an advance rate of 6½ feet per shift or 154 shifts per 1000 feet. The 5-entry system used in subplan development is based primarily on its flexibility in carrying sufficient ventilating air while also being useful for haulage. The particular criteria or characteristics which call for alternative use of a 10- or 5-entry system are based on the importance of the system to the mine in general, the length of time it must be available, the number of panels being served by it and the amount of ventilating air it must carry. In the planned multiple seam mines a 10-entry system may be used in an upper seam where only a lower seam requires it. Such an upper main will be required to maintain super position.

Entry use in a 5-entry subplan will be similar to that for the 10-entry. Two side entries will be used for intake air, the other two side entries will be used for return air. The center entry will be used for both belt and track/supply haulage.

The 3-entry longwall development system is a rather uncommon feature in American longwall mining, but it has found growing use thus far. Figure 3-8 illustrates typical 3-entry section development as it is currently being conducted at a UPCC mine in central Utah. Pillar sizes



TWO ENTRY DEVELOPMENT



FIVE ENTRY DEVELOPMENT

LEGEND

CHECK CURTAIN — LINE BRATTICE

TRAVEL CURTAIN — STOPPING

GARFIELD PROJECT
GARFIELD COUNTY, UTAH

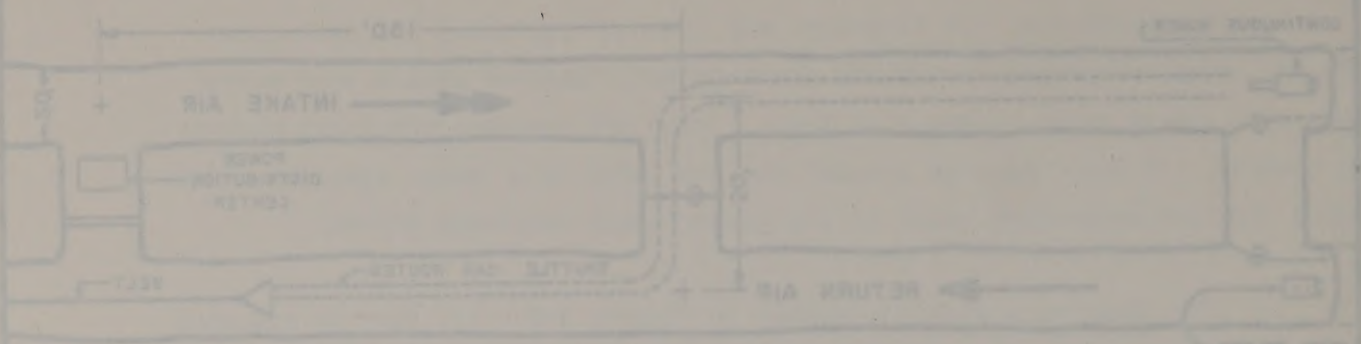
SUBMAIN AND PANEL DEVELOPMENT BY CONTINUOUS MINERS

UTAH POWER & LIGHT COMPANY

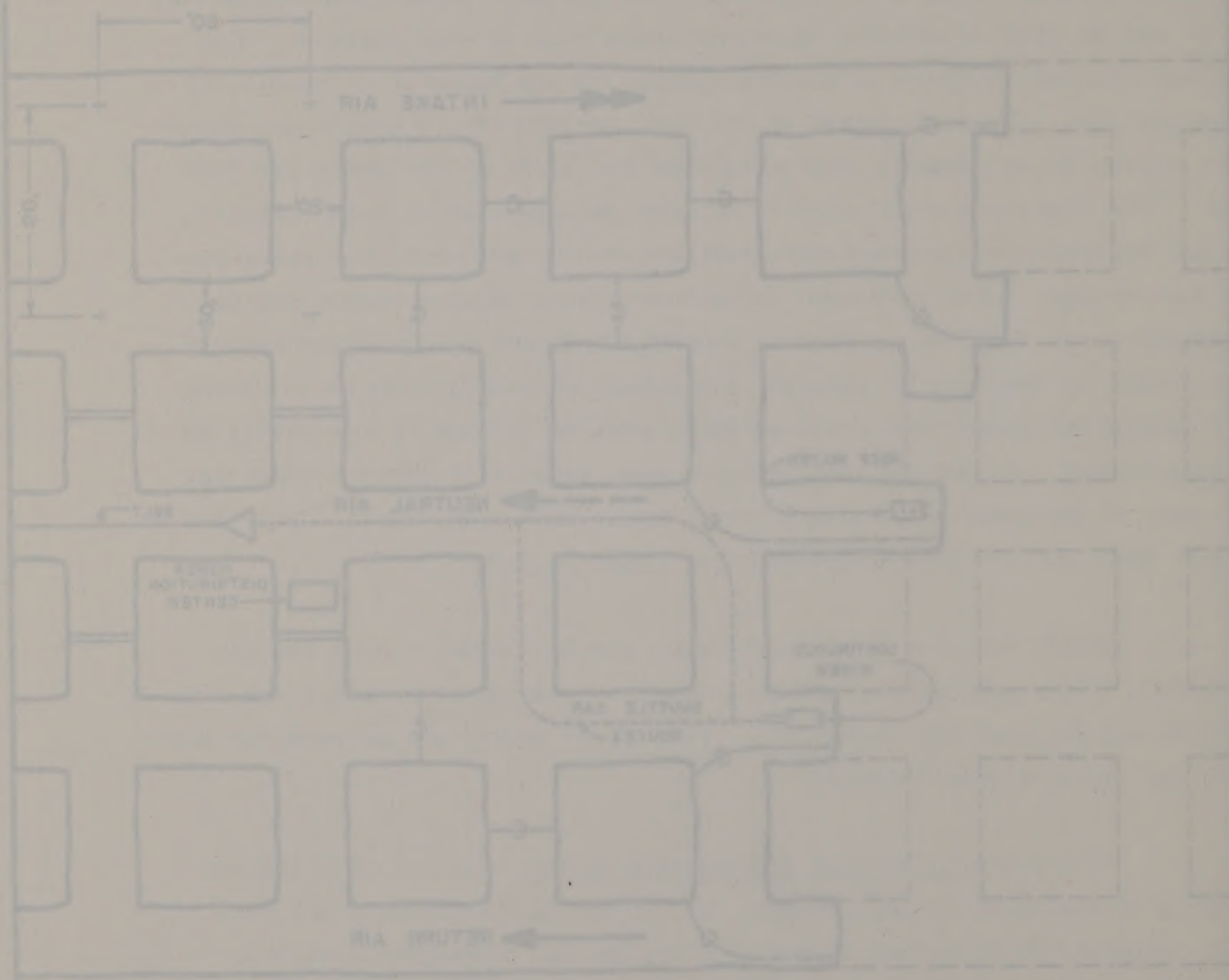
SCALE: 1" = 50'

FIG. NO.: 3-8

DATE: AUGUST 20, 1979



TWO ENTRY DEVELOPMENT



FIVE ENTRY DEVELOPMENT

GARFIELD PROJECT	
SUBMAIN AND PANEL DEVELOPMENT BY CONTINUOUS MINERS	
SCALE 1"=50'	UTAH POWER & LIGHT COMPANY

and crosscut spacing are considerably different from other entry patterns: 30 foot-width of pillars, and 150 foot between crosscuts is one accepted practice currently in use in Utah.

Operationally, the miner and the bolter simply exchange places when a cut is finished. Coal loadout is done in the return where the belt is located, a variance from standard practice which is acceptable as long as permissible (schedule 31) equipment is used. Ventilation in the return entry is accomplished by running a line brattice from the face to the belt hood. This enables the shuttle car operators to work in fresh air conditions while traveling in the returns. Rates of advance are determined similarly to those for other entry systems and are based on 17 tons per foot which yields 20 feet per shift or 50 shifts per 1000 feet.

The pillars have been designed to withstand loss of one entry when used as a headgate, then crush out after being used as a tailgate. Alternatively, the pillars may wedge up through the roof but the goal is to relieve downward transmission of pressures to the pillars in seams below. Alternatively, the pillars in the lower seams may be off-set slightly to allow a relief of the pressure. Whatever the case, super position should either be exactly maintained or the off-set should be equal to the pillar width.

At intervals of three to five panels, 200 foot thick barriers are left between panels to act as firebreaks. The concept is based on the possibility that spontaneous combustion may occur in the gob and that these firebreaks are necessary to contain air inflow and rate of burn. Mining the panels between these firebreaks must be sequential and should be done in accordance with an overall planned direction for the mine.

All of the above described entry development will be performed within a mine by six to ten complements of continuous mining production sections. Each complement would consist of one continuous miner, two shuttle cars, a feeder breaker bin, and a panel conveyor belt to handle

the coal, a roof bolter to enable the required roof control measures to be installed, a diesel scoop for supply and cleanup assistance, and all the usual accessories. These items are detailed, with costs, in Table 3-10; the unit cost for a continuous miner section is used later in Table 3-12 to determine production equipment costs for each mine.

Longwall Extraction of Panels

Figure 3-9 shows a typical longwall panel as it will be used in these mines. The central idea in longwall mining is to set up supports along the short dimension of a large rectangular block of coal and proceed to extract all of the coal out of the panel in 2 to 2½ foot cuts. The dimensions of the panels outlined in these mines are 500 foot nominal width with lengths ranging from 5,200 feet down to 2,400 feet. In the future the use of improved designs and materials in the face conveyor may allow longwall blocks to become 1,000 feet wide by up to 10,000 feet long. Use of blocks this large would reduce the use of continuous miners and enhance the percentage of recovery. Table 3-11 is a listing of the necessary longwall face components, equipment pieces, and their costs.

Production rates from longwall faces operating under conditions similar to those planned for these mines will range from 1,000 to over 4,000 tons per shift. A production figure representative of current new longwall faces would range around 2,000 tons per shift. For this study, 1,500 tons per shift was chosen as a very conservative estimate. It should be noted that higher production rates imply an increase in tons per man shift, reduction in the required numbers of longwall units, and a consequent reduction in the labor force. Using 1,500 tons per unit shift, with each unit working two shifts per day, one and a half years and required to mine a panel 5,000 feet long by 500 feet wide with an average seam height of 10 feet.

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Production rates from longwall faces operating under conditions similar to those planned for these mines will range from 1,000 to over 3,000 tons per shift. A production figure representative of current new longwall faces would range around 2,000 tons per shift. For this study, 1,500 tons per shift was chosen as a very conservative estimate. It should be noted that higher production rates imply an increase in tons per man shift, reduction in the required numbers of longwall units, and a consequent reduction in the labor force. Using 1,500 tons per unit shift, with each unit working two shifts per day, one and a half years and required to mine a panel 5,000 feet long by 500 feet wide with an average seam height of 10 feet.

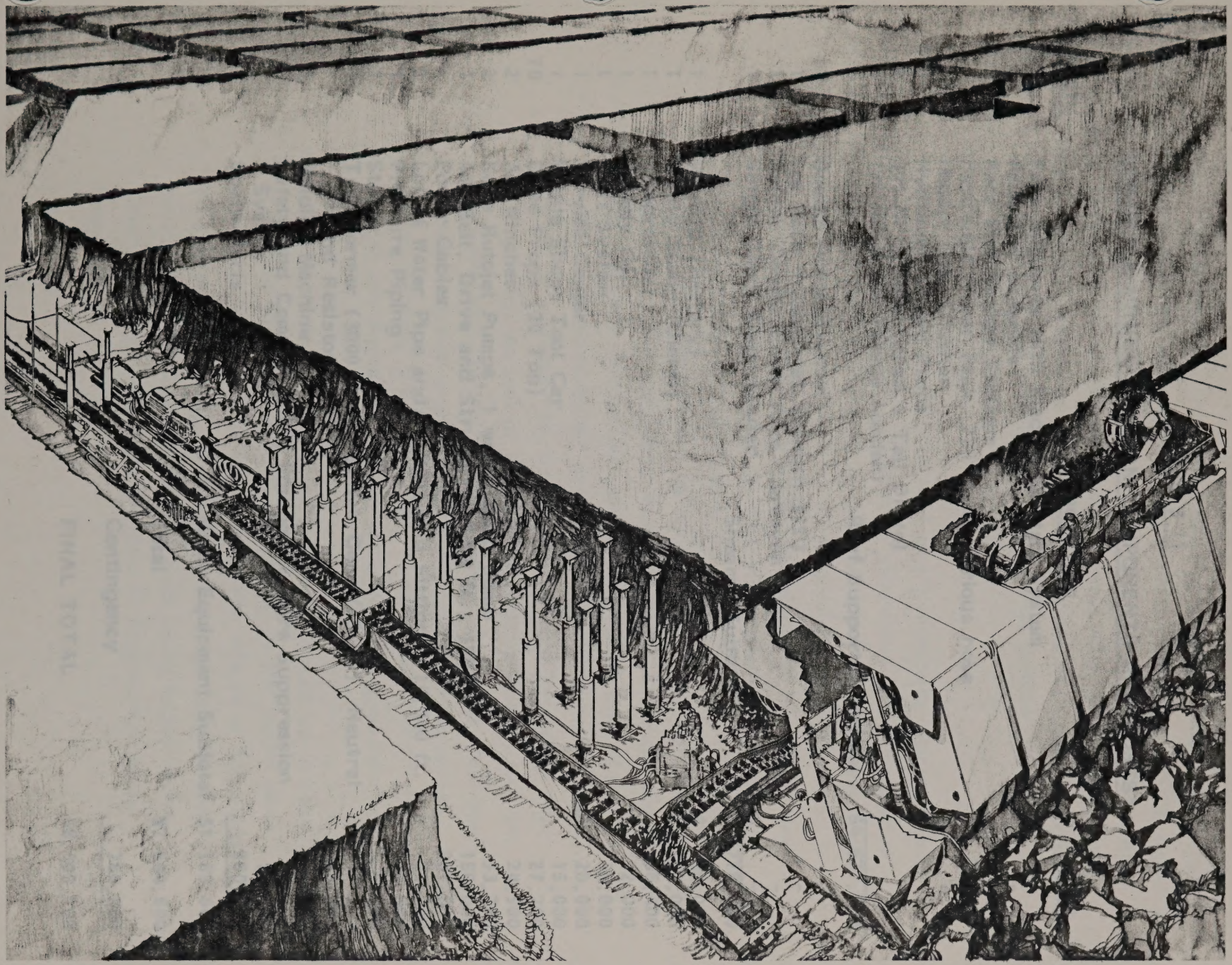


FIGURE 3-9: Typical Longwall Panel

FIGURE 2-3 Typical roadway layout

TABLE 3-11

LONGWALL MINING SECTION COMPONENTS AND COSTS*

Quantity	Equipment Item	Cost
117	Face Supports with Electric Control System	-
6	Face End supports	-
2	Face End Anchor Stations	-
1	Hydraulic Power Pack, 700 ft. of various hose	-
1	Impact Coal Crusher	-
100	Hydraulic Single Props	-
2	300 HP Motors (Head & Tail Drive)	-
1	125 HP Motor (Stage Loader)	-
	Roof Supports Subtotal	$4,677,000
1	Double Drum Shearer	-
1	Lot Face Accessories (Length 575')	-
1	Double Center Chain Face Conveyor	-
1	Single Chain Stage Loader	-
	Shearer Subtotal	$1,778,000
1	Mantrip (Battery)	$ 80,000
1	Utility Scoop (Diesel)	120,000
1	Compressor	36,400
1	Supply Car	22,000
1	Dust System	36,000
1	Service Center	20,000
1	Mobile Shop, Tool Car	15,000
70	Duke Props (20 Ton)	27,700
2	Air Winches	24,900
2	Kobe Rotojet Pumps, 1 water sump pump	53,700
1	48" Belt, Drive and Structure, 5000 ft., 150 HP	186,000
-	Power Cables	207,800
1	Lot of Water Pipe and/or Hose and Fittings, 10,000 ft.	34,200
1	Pressure Piping	63,800
1	Stoper	4,300
1	Transformer (3000 Kva), Switch Gear (15 Kv), Neutral Ground Resistor	43,800
1	Welding Machine	2,000
-	Safety and Communication Equipment, Fire Suppression System	12,000
-	Spare Parts	200,000
	Misc. Equipment Subtotal	$1,189,600
	Total	$7,644,600
	Contingency	764,500
	FINAL TOTAL	$8,409,100

The 10-foot average criteria is used even though seam thicknesses range up to 24 feet. The state of the art of multilift mining, or any other technique used for thick seam mining is not advanced in the United States. The primary limitations to mining more than 10-feet high with a single lift are that face abutment pressures cause spalling of the face coal. Fallen chunks of coal may plug the conveyor. Further, at greater height the face could blow out, causing injury to workers.

Although not particularly critical to a preliminary plan and cost estimate such as this, there are a number of principles which should be followed in using longwall techniques in general and in multiple seam mining in particular:

- As previously stated, the main development entries should be exactly superimposed so that overburden pressures are focussed vertically downward.
- Mining sequences should be consistent. For these mines panels will be sequentially mined. In the direction away from the main entries out away from the portals, the intent is to maintain an even lowering of the overburden. Mining of the uppermost seam will be most crucial in that the greatest amount of overburden will be broken and the most unstable conditions could exist. Once the rock has broken, internal stress components will be released probably relieving the forces that tend to cause rolls. Panels of coal will definitely not be mined out of sequence at these mines.
- Longwall mining in any subjacent seam should be at least two to three panels behind the mining in the upper seam. The purpose in this is to minimize the effects of the lower mining on the upper. Subsidence from the lower mining could affect the bottom of the panels being mined above. Clearly a geotechnical optimization of angles of draw is required.

- It may be necessary to make panels narrower in each subjacent seam so that the firebreaks can be made wider and more stable. Doing this would depend on the ascertained need for the firebreaks and the stability of existing designs.
- Production should be spread over a large area. Only one longwall unit should be used in any particular section of the mine, to avoid having two panels feeding into one submain belt but from opposite sides. This will increase the need for belting but will result in more orderly and efficient mining. Because of their negligible impact on the overburden, continuous miner units might be placed anywhere as required.

Table 3-11 lists the equipment components of a typical longwall section, divided into three groups: the roof supports or shields, the coal production units - shearer and conveyor, and all the miscellaneous support equipment. This unit cost for a typical longwall section is used in Table 3-12 in the determination of total production equipment costs in each mine.

TABLE 3-12

CAPITAL COSTS OF PRODUCTION UNITS

(Millions of Dollars)

Item	Expected Life	Cost/Unit	Budget Amount										
			No.	Mine #1	No.	Mine #2	No.	Mine #3	No.	Mine #4	No.	Total	
Longwall Mining Section	10 years	8.41	3	25.23	5	42.05	5	42.05	5	42.05	18	151.38	
											Spares*	23.70	(16%)
												175.08	
Continuous Mining Section	5 years	1.73	6	10.38	10	17.33	10	17.33	10	17.33	36	62.37	
	Mine Totals			35.61		59.38		59.38		59.38	Spares**	15.26	(24%)
												77.63	
												$252.71 MM	

*Longwall Spares Include:

1 Face Conveyor per 1 Longwall	$ 700,000 x 18	=	12.6 MM
6 Shields per 1 Longwall	$ 210,000 x 18	=	3.78
20% of Accessories per 1 Longwall	$ 240,000 x 18	=	4.32
1 Shearer per 6 Longwalls	$1,000,000 x 3	=	3.00
			$23.70 MM

**Continuous Miner Spares Include:

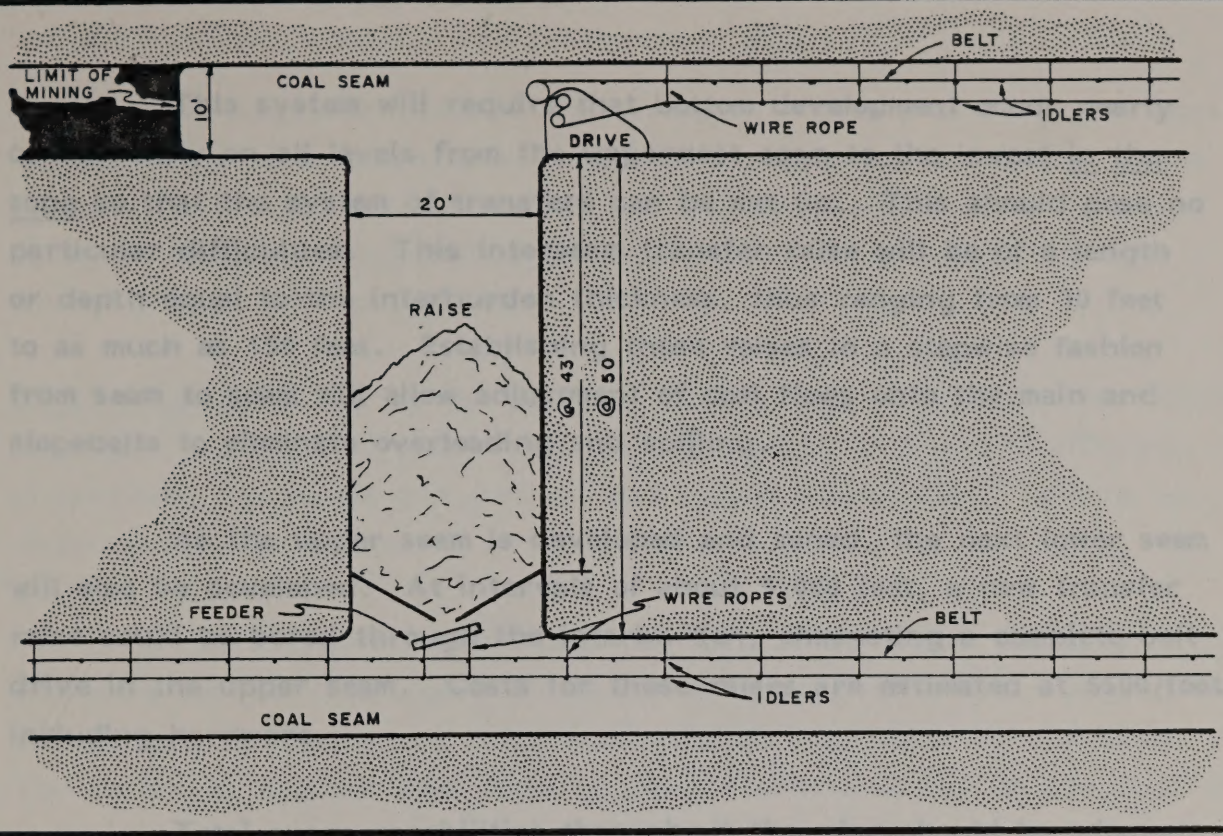
1 Continuous Miner per 5 Planned	$410,000 x 7	=	2.87 MM
1 Shuttle Car per 4 Planned	$270,000 x 18	=	4.86
1 Roof Bolter per 5 Planned	$132,000 x 7	=	0.92
20% of Accessories	$184,000 x 36	=	6.61
			$15.26 MM

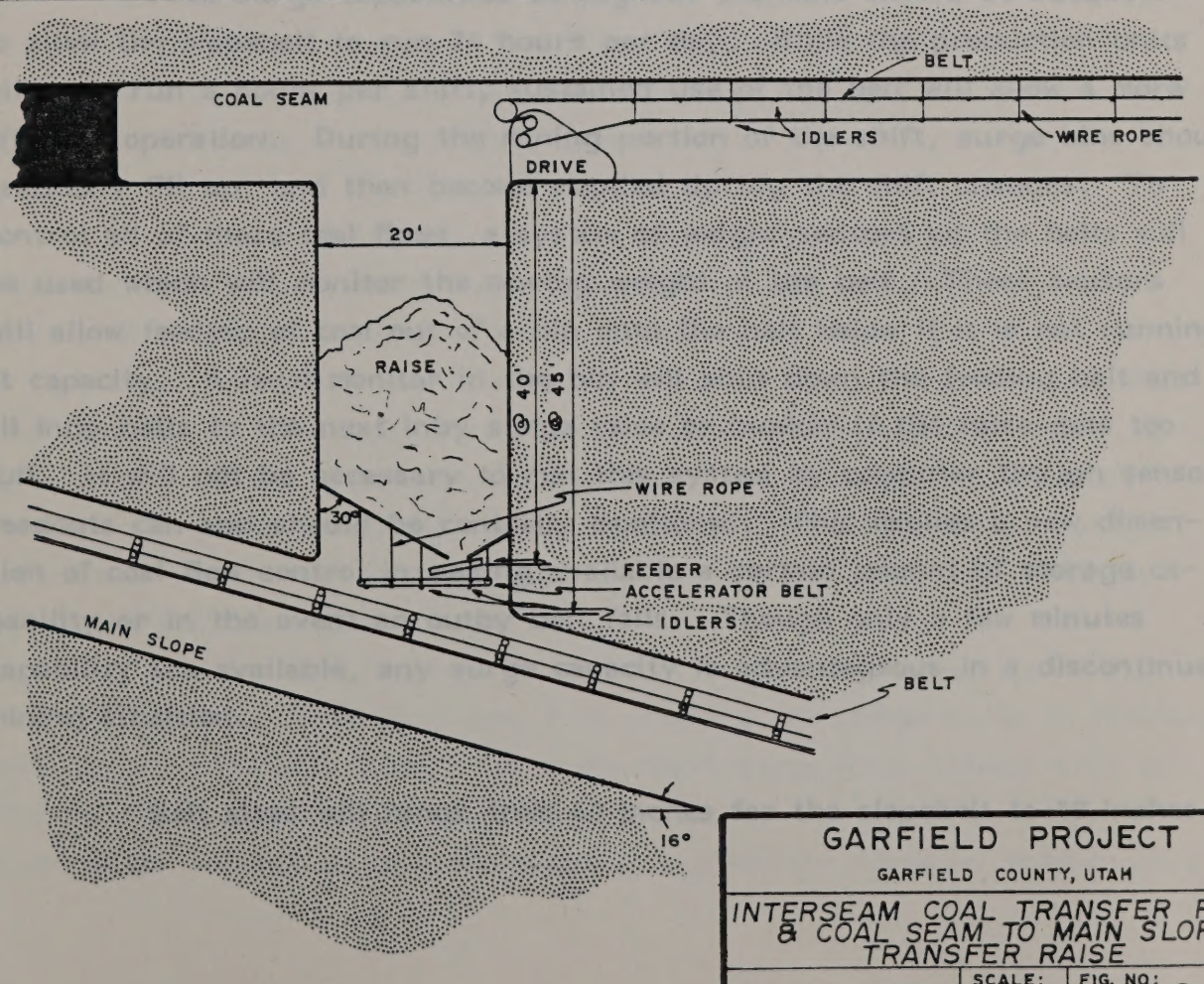
Three separate transportation systems must be set up for personnel transport, coal haulage and supply haulage.

Coal haulage will be exclusively by belt from the stage loader or feeder-breaker to the loadout facility on the surface. Surges that come from batch haulage (shuttlecars) interfacing with the continuous system or from unexpected surges from the longwall faces can be a major problem, but are considered manageable with the use of surge bins and bunkers. These will allow flow controls at any point where a belt capable of handling the nominally sustained flow is likely to be overloaded. Surge bins can be constructed in raises in the interburden between seams, allowing lowering of coal from seam to seam. Loadout into the slope belt will take place from only the basal seam in any of the coal zones, with a maximum of three loading points (one for each zone) onto the slope belt. This will be modified if it is found that interburden thicknesses allow transfer between zones. The reason for minimizing dump points is the difficulty and expense involved in setting up a dump point on a steeply inclined belt.

The actual configuration of the slope/raise interface is fairly simple. The slope will be in one of the barrier pillars adjacent to a 4-entry system while the coal transfer raise would be located beneath an entry room, so a sloped bottom in the raise will deliver coal to the feeder/accelerator belt. All coal from all seam levels would be transferred to the slopebelt by way of such a transfer-raise surge bin. Surge bin capacities will be approximately 630 tons for every 50 feet of length of 20 ft. diameter raise. Though Figure 3-10 indicates a length of 40 feet, this is only nominal and would actually depend on the desired surge capacity.

Figure 3-10 gives an impression of possible coal movement as it is passed through successive raise/surge bins to finally arrive at the slopebelt.





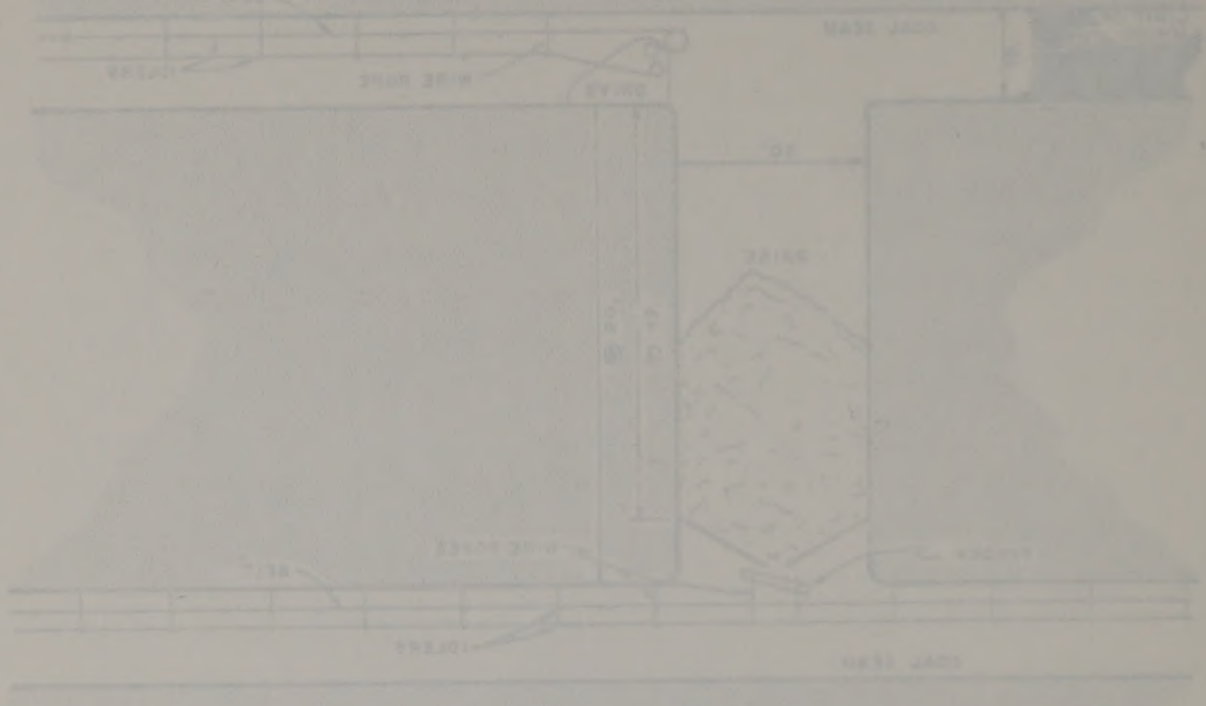
—CROSS SECTIONAL VIEWS—

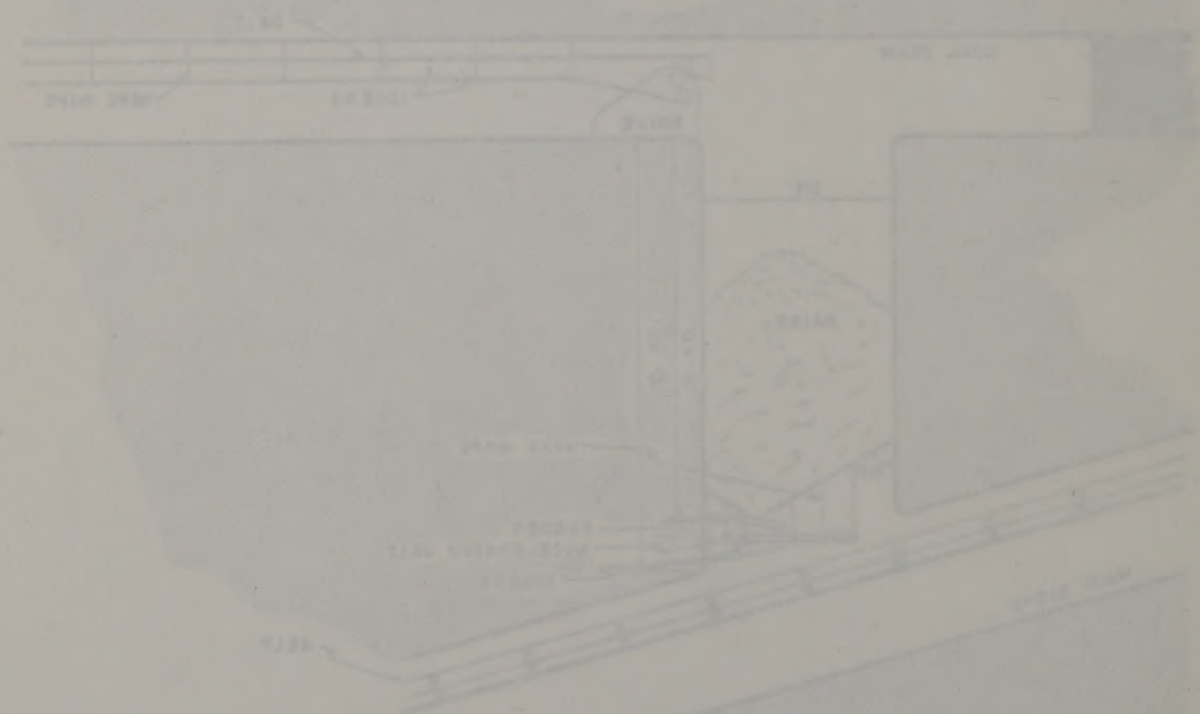
GARFIELD PROJECT

GARFIELD COUNTY, UTAH

INTERSEAM COAL TRANSFER RAISE & COAL SEAM TO MAIN SLOPE TRANSFER RAISE

UTAH POWER & LIGHT COMPANY	SCALE: NONE	FIG. NO: 3-10	DATE: AUGUST 20, 1979





This system will require that bottom development occur nearly concurrently on all levels from the uppermost seam to the lowest in the zone so that the system of transfers can be set up. This should pose no particular difficulties. This interseam transfer raise will be of a length or depth equal to the interburden thickness, thus ranging from 30 feet to as much as 150 feet. Establishing these raises in a stepwise fashion from seam to seam will allow adjustment of coal flows onto the main and slopebelts to eliminate overloading and spilling.

As the upper seam is developed and mined, the next lower seam will also be developed. At intervals of about 5,000 feet, a new transfer raise could be bored through the interburden, eliminating a complete belt drive in the upper seam. Costs for these raises are estimated at $500/foot, including hardware.

Total surge capabilities throughout the mine should be adequate to allow the slopebelt to run 24 hours per day. While the production units will only run 5 hours per shift, sustained use of the belt will allow a more efficient operation. During the mining portion of the shift, surge bins should gradually fill up, and then become emptied during the shift changes. To control all of these coal flows, a system of weight sensors on the belts will be used which will monitor the moving weight of the belt. These sensors will allow feeding of coal out of a bin onto the belt below if it is not running at capacity. A level monitor in the bin will shut down the feeding belt and all inby belts to the next inby surge raise or bunker if the raise gets too full. It will not be necessary to run this system by computer though sensor readouts can and should be centrally monitored. This implies a new dimension of coal flow control in making available a certain amount of storage capability or in the event an outby belt fails. Though only a few minutes capability are available, any surge capacity is advantageous in a discontinued mining situation.

Belt sizes will range from 60 inches for the slopebelt to 48 inches

for all of the individual panel or section belts. Table 3-13 shows the sizes of belts to be used, all necessary ancillary equipment, the capacity of the belt and its probable function within the haulage system. Table 3-14 is a listing of equipment expected to be required for the entire coal haulage system of each mine and its cost.

Supply haulage will be by two separate modes - combination rail and rubber-tired cars. The rail system should be high quality to maximize efficiency in servicing the panels and sections and should be capable of holding up under weights as high as 55 to 60 tons per car.

The system will begin with the hoist at the top of the slope which should be capable of bringing 75-100 ton loads up a 16° slope. Staging areas will be built at the level of every minable seam intercepted by the slope. The The switch for a given level must be placed sufficiently far up the slope that the turnout can be made from the slope, the track will be curved back to a straight downslope direction, then the transition is made from the slope to the horizontal. The actual distance of the switch up the slope will be a function of the size of switch frog used.

Supply haulage will utilize 20-ton 8-wheel supply cars hauled by diesel locomotives of various sizes. No trolley wires are planned in these mines. Haulage track will be 60 lb./yd. rail in the mains and 60 lb./yd. rail in the submains. The rail will only be run in the mains and submains with small sidings at the mouth of each section used for staging the supply cars. The 2-entry system is not amenable to use of rail because of the severely congested conditions. Diesel scoops will be used to transfer the supplies from these cars to the working face. A list of all of the supply haulage equipment and costs is shown in Table 3-15.

As mentioned in Section 3.21, a few of the seams occur in isolated pods some distance from the slope. To reach these seams, ramps must be constructed from the seam immediately below. At the top of the ramp is a hoist similar to that serving the slope, but generally somewhat smaller since

TABLE 3-13

REPRESENTATIVE MINE CONVEYOR BELT CAPACITIES

Belt Size (In.)	Application	Speed (FPM)	35° Idlers Max. Required Capacity (TPH)	35° Idlers Peak (TPH) Capacity	Capability
48	Panel or Submain Belt	250	700	713	2CM or 1LW
		250	500	713	
		300	850	855	1CM & 1LW
		400	1,050	1,140	3CM
		450	1,200	1,283	2CM & 1LW
		500	1,400	1,425	4CM
		600	1,700	1,710	2CM & 2LW
60	Main Belt	100	347	480	2CM & 1LW
		150	694	720	4CM & 2LW
		250	1,041	1,200	6CM & 3LW
		300	1,388	1,440	8CM & 4LW
	Slope Belt	300	1,388	1,440	8CM & 4LW

Assumptions:

1 C.M. Production = 330 T/S $\approx$ 51 TPH
Peak Instantaneous Load on Belt = 350 TPH
(This is ratio feeder setting. It allows 1.37 min. for S.C. to off-load 8T).

1 LW Production = 1500 T/S $\approx$ 231 TPH
Peak Instantaneous Load on Belt = 500 TPH
(This is setting on ratio feeder. It allows for surge capacity of approximately 2x).

Surge Bins located at loading points between seams allow design of main and slope belts for average capacity.

TABLE 5-13

REPRESENTATIVE MINE CONVEYOR BELT CAPACITIES

Belt Size (in.)	Application	Speed (FPM)	Max. Required Capacity (TPH)	Peak (TPH) Capacity	Capability
42	Panel or Submain Belt	250	700	713	2CM or 1LW
		350	980	855	1CM & 1LW
		500	1,050	1,140	3CM
		650	1,200	1,283	2CM & 1LW
		500	1,400	1,425	4CM
		600	1,700	1,710	3CM & 2LW
60	Main Belt	100	347	480	2CM & 1LW
		150	634	720	4CM & 2LW
		250	1,041	1,200	6CM & 3LW
		300	1,388	1,440	8CM & 4LW
	Slope Belt	300	1,388	1,440	8CM & 4LW

TABLE 3-14

COAL HAULAGE SYSTEM COSTS

Item	Expected Life	Cost/Item (Dollars)	Mine #1	Mine #2	Mine #3	Mine #4
			(Thousands of Dollars)			
Slope Belt - 60" - 16° incline						
Flexsteel 2000 p.i.w.	10 years	$55/ft.	$ 330	$ -	$ -	$ -
Flexsteel 3300 p.i.w.	10 years	$70/ft.	-	-	-	476
Flexsteel 4000 p.i.w.	10 years	$90/ft.	-	774	738	-
Belt Structure - 35° through 4' idler spacing - 8' return idler spacing	20 years	$318/ft.	954	1,367	1,304	1,177
Belt Drive, Tailpiece, Counterweight	20 years	$130/HP	156	312	321	260
Main Belt - 60"						
Belt - 600 p.i.w.	10 years	$23/ft.	1,472	1,150	1,012	828
Belt Structure	20 years	$35/ft.	1,120	875	770	630
Belt Drive, Take-up, & Tailpiece	20 years	$10/ft.	350	250	250	200
Submain Belt - (48" section belt in each mining unit)						
Belt - 300 p.i.w.	10 years	$17/ft.	126	1,020	918	1,360
Belt Structure	20 years	$22/ft.	82	660	600	880
Belt Drive, Take-up, & Tailpiece	20 years	$8/ft.	40	240	240	320
Transfer Raises/Surge - 20 ft., Unlined 60' average	40 years	$500/ft.	180	270	270	90
Surge Transfer Equipment	40 years	$10,000	30	60	60	30
Main Slope Transfer Equipment	40 years	$40,000	120	120	120	80
Accelerator Belt onto Slope Belt	20 years	$50,000	150	150	150	100
Belt Haulage Mine Totals			$5,110	$7,248	$6,753	$6,431

TOTAL $26,037

TABLE 3-15

SUPPLY AND PERSONNEL TRANSPORT COSTS

(Thousands of Dollars)

Item	Expected Life	Cost/Unit	Anticipated Amount			
			Mine #1	Mine #2	Mine #3	Mine #4
Locomotives - 20 ton Diesel	20 years	$125	$ 625	$ 750	$ 750	$ 750
Locomotives - 15 ton Diesel	20 years	105	630	735	735	735
Trackman/Beltman - 15 ton Battery	10 years	60	240	240	240	240
Maintenance - 15 ton Battery	10 years	60	360	360	360	360
Safety & Supervisory - 10 ton Battery	10 years	50	250	250	250	250
Supply Cars - 20 ton Capacity (not on section)	40 years	22	528	880	880	880
Longwall Shield Trailer	20 years	26.5	265	265	265	265
Low Boxes (Equipment Movers)	20 years	32	96	96	96	96
Longwall Scoop - 20 ton Capacity	10 years	135	270	270	270	270
Bulk Dusters - 20 ton Capacity	20 years	45	90	90	90	90
Track Mounted Bolter	10 years	85	170	170	170	170
Ballast Cars	20 years	15	45	45	45	45
Track Cleaner	10 years	95	95	95	95	95
Track - 60#/yd with accessories	20 years	$36/ft.	2,520	4,320	4,824	4,320
Personnel Elevators (one each mine)	40 years	300	300	300	300	300
			$6,484	$8,866	$9,370	$8,866

TOTAL $33,586

hoisting speed can be slower and the maximum ramp length is 700 to 800 feet compared to around 4,000 feet for the slope.

The amounts and kinds of supplies requried by any single section on a particular day can vary greatly. Use of a two or three car supply train which will be loaded with the complete variety of supplies normally required by a particular type of production unit is envisioned. Supplies will not be off loaded from the supply cars, rather the cars will remain in place until the supplies have been depleted at which time a refilled two or three unit supply train will be brought in and exhanged. These extra supply cars represent a large capital expenditure, but they are jusitifed by the alternative of losses through rust, neglect, or wastage which can be major expenses, and the manpower needed to unload every car. Similarly, every attempt will be made to protect, palletize and maintain the supplies in the surface yard.

Personnel transportation will be by battery prowered mantrip on the seam levels. Crews will travel by elevator from the changehouse to a point near the slope bottom. Mantrips will than be used to carry them from the slope bottom to the supply track switchout at which point either a small mantrip trailer hauled by the scoop can be used or, the crew may walk to the faces. The equipment required for personnel transport and costs are tabulated in Table 3-15.

3.25 Underground Services

Any underground mining venture requires a number of ancillary systems or operations. The necessity for these systems may originate in legal mandate, in the requirements of equipment operation, or for employee morale. Principal among these services are ventilation, power, water, rock dust, and communications systems.

Ventilation of these mines will generally be far more complex than for a single seam mine due primarily to the intricate 3-dimensional nature of the operation. The primary concerns in proper ventilation of any mine are preventing leakage or short circuiting, diluting and expelling noxious and toxic gases and meeting the health and safety requirements of the law. The legal requirements are generally exceeded since most responsible companies try to get far more air to the face than the legal minimums.

For this mining plan, the ventilation requirement has been set at 75,000 c.f.m. for the continuous miner development sections. The allocation for a longwall panel has been set at 200,000 c.f.m. This will allow for bleed-off of gas through the gob and short-circuiting into seams above through the broken interburden. Supplying this much air will require at least three fans at the larger mines of 1,000 HP each capable of supplying 500,000 c.f.m. each. Since the lateral extent of these mines is not that great, one fan will be placed near the portal and the other two will be separated as seam geometry, topography, and shaft construction costs allow.

The mains (ten entries) will constitute the principal airways with a minimum of three entries each for intake and return air with more available if needed. The submains (five entries) will allow 2 entries each for intake and return air. This should be sufficient since no more than two longwall units and two continuous miner units will be working inby a particular submain. The figures illustrating development entry systems in Section 3.22 detail the air flow at the face.

Ventilation airways will be maintained between seams to aid in equalizing air flows and in distributing air to where it is needed. These connections will be by way of the slope which constitutes a fresh air intake, air shafts and auxiliary ventilation shafts which will connect with every seam, as well as interseam air raises located near ramps and other strategic locations. A great deal of use will be made of these interseam raises to take full advantage of all available airway capacity. These air raises are

similar in construction, but separate in function from the interseam coal raises. If production is maintained in two seams with development in a third, the lowest, set of entries will probably be underutilized. Use of ventilation raises will reduce the air velocity in any one of the levels and provide a more balanced air flow.

Ventilation requirements depend to a large extent on the gasiness of the seams and overlying strata. Many coals are subject to spontaneous combustion when broken and left in a wet place such as the gob. Various measures can be taken to prevent these fires. Use of bleeders to draw cooling air into the gob may only fan the fire. Total sealing of the gob on three sides will allow the gas to concentrate past its explosive point. Some mines which have had these problems have discontinued use of bleeders and have managed thereby to aid in face ventilation as well as reducing the chance of fanning the fire.

Another technique practiced in Europe involves forming a packwall in the gob by shooting suitable material right in back of the shields. The tendency to spontaneously combust has not been fully assessed for Kaiparowits coal so the need for bleeders or packwalls in the gob cannot be stated.

There are state minimum quantity and quality requirements in areas on haulage ways where diesels are used. For Utah, whose requirements supersede federal limits, 150 c.f.m. of fresh air is required for every brake horsepower with scrubbed exhaust quality standards set at .5% CO_2, .01% CO and .0025% NO_x.

Leakage is a problem at nearly any mine. These mines will be especially subject to leakage problems because in most cases longwalls will be mining below previously worked areas. Typically, in mines using good stopping construction and sealing practices, an amount of air equal to 20% of the air reaching the end of a long segment will be lost through leakage. Also, typically, 40,000 c.f.m. are required at the mouth of a panel development to obtain 10,000 c.f.m. in the last open crosscut. Many mines currently

use steel temporary stoppings in their panels which can push that initial requirement to 60,000 c.f.m. Paralleling this, regulation of the air is complicated by fans pulling air in an upper gob area short-circuiting air from a lower production level. In this situation, the entire broken roof area becomes a bleeder.

The most widely accepted method of dealing with the ventilation problem is to use larger capacity fans and increase the usual air allotments to longwall panels and continuous miner sections. Miner sections working in development are not expected to use a great deal more air than usual, since no retreat mining is done and the roof reamains competent. Table 3-16 details the costs for individual components in a mine ventilation system.

The power requirements for the underground complex will be met by a 13.8 KV line dropped down a borehole or an airshaft to the mine bottom. The 13.8 KV line will run through a switch and transformer unit at each of the various seam bottoms to supply 4160 volts to the seam bottom shop and for further distribution to the mine. The 4160 volt line will be fed subsequently into transformers to reduce it to 440 volts at each belt drive including the accelerator belts for loading the slopebelt. This 4160 volt line will also proceed throughout the mine to each working section or panel where it will go through a transformer for reduction to 950 volt for use by certain larger equipment pieces and 440 volt for the roof bolter, shuttle cars, feeder breaker, and stage loader. Table 3-16 contains a listing of the material and unit costs necessary for the mine electrical system.

Water will be needed underground primarily for dust suppression and equipment cooling (continuous miner). The quality of the water used must also be such that water sprays will not be clogged or corroded. There are two possible sources fo the necessary water in the Garfield Project Area: ground water inflow into the mine from coal zone strata, or tapping by drilling of the Navajo Sandstone aquifer several thousand feet below. As

previously stated, there is little specific information on the hydrology of the mine area so preliminary attempts to estimate ground water inflows will be rough. Use of inflow water in the mine will require that it be gathered in sumps of large enough total capacity that sediment can be precipitated. Following this the water must be treated, to prevent corrosion, filtered three times, and finally piped to the areas where it is needed. Additional water, if needed, could be pumped up from the Navajo sandstone and used with little or no treatment or filtering. Water may also be gathered as surface runoff in the sediment ponds in the surface mine facility areas.

The amount of water required for underground mine operations is principally a function of the amount of coal mined and the amount of time the mining operations are functioning because of the use of water in dust suppression systems. Usually, adequate dust suppression underground requires enough moisture to raise the weight of coal mined by 1.5%. On that basis, a longwall panel will require 45 tons or 10,800 gallons of water per day while a continuous miner unit will require 9 tons or 2160 gallons of water per day. The total dust suppression water requirement is 324 tons or 77,760 gallons per day of water for the larger mines and 215 tons or 51,600 gallons per day at Mine No. 1.

Distribution of the water will be by a 12-inch diameter fiberglass pipe laid down the slope. At each seam, a 6-inch lateral line will be laid down which will follow the main entries. Pipes on submains and development entries will be 4 inches in diameter. This water will be used only to augment the supply of water from the sump. The plan will be to place the highest priority on use of the sump water followed by surface waste water and runoff, and last choice will be the use of potable waters. Water from the surface will not need pumping because of the obvious high heads developed down slope. Booster pumps will be placed in the line once operations are far enough from the portal that line losses are a significant factor.

Table 3-16 lists the equipment needed, the expected life, and the unit price for a mine water distribution system.

The communications network for the mine will be similar to other mines in having a radio mounted in each producing section and on most pieces of moving equipment such as locomotives and jeeps. Supplementing this will be a standard wired underground telephone system installed at each production section and other strategic locations, such as major belt transfers and at the shops.

A parallel system will be a wire communications network connecting surge bin loading sensors with the central complex communication/computer system. The purpose of this system will be to provide continuous information and control over the loading of the various belts throughout the mine and the relative quantities in the surge bins. Automated electro-mechanical gates on these raise bins as well as automated emergency control over belts will also be provided. These systems are not meant to totally supplant individual operator control but provide a broader based control over mine operations.

Rock dust distribution is required by Federal and State law to reduce the possibility of coal dust explosion by diluting or covering any coal exposures or coal dust residues with inert limestone rock dust. The first concern is in maintaining good housekeeping to reduce the available coal dust. On a percentage basis, the rock dust must constitute a minimum of 65% of any loose material.

Distribution of rock dust will involve a three-level system. Using a pneumatic system primarily, with secondary use of track-mounted "pod dusters" for the parts of the mine further from the bottom. Bagged rock dust will also be used in those situations where dust is required in an emergency stockpile or other spot situations.

The pneumatic system for each mine will be comprised of a 150 ton dust bin on the surface, directly beneath will be a six inch lined borehole extending down to the lowermost seam. At each producing seam level there will be pressurized ASME-standard dust tank equipped with a compressor/

fluidizer. The dust will be conveyed through a $3\frac{1}{2}$ inch fiberglass pipe out into the mine. The projected capability of a single compressor is 2500 feet of pipe length. As planned, a 10,000 foot system will have three booster pods with compressors at 2500 foot intervals. The longwall development and production panels within 10,000 feet of the bottom will be serviced directly by the pneumatic system.

The track-mounted pods will be loaded at any point along the pneumatic line which has a capability of about 80 pounds per minute or at the pressurized vessel at the borehole. Table 3-16 includes a summation of the capital expenditures for the dust system.

CAPITAL COST ESTIMATE
UNDERGROUND UTILITY SYSTEMS - VENTILATION, POWER, WATER, ROCK DUST, COMMUNICATIONS

Category/Item	Expected Life	Cost/Item (Dollars)	Mine #1	Mine #2	Mine #3	Mine #4	
			(Thousands of Dollars)				
Ventilation							
500,000 cfm fan	20 years	$500,000	$1,500	$1,500	$1,500	$1,500	
Overcasts, 10 per mining unit	40 years	6,000	540	900	900	900	
		Subtotal	$2,040	$2,400	$2,400	$2,400	$ 9,240
Electrical							
Access Borehole, lined	40 years	$50,000	50	50	50	50	
Cable, SHD-6C, 15KV	20 years	$8.50/ft.	10	10	10	10	
8KV	20 years	$18.00/ft.	643	990	880	900	
Main Transformer & Switchgear (13,800-4, 160 Vg one each seam)	20 years	$50,000	250	300	250	150	
Belt Power Center	20 years	$15,000	120	165	165	180	
		Subtotal	$1,073	$1,515	$1,355	$1,290	$ 5,233
Water							
Slope Main, 8" steel	20 years	$22.00/ft.	66	95	90	75	
Level Main, 6" fiberglass	10 years	$5.50/ft.	176	138	120	100	
Level Submain, 4" fiberglass	10 years	$3.50/ft.	73	105	95	140	
Auxiliary pump	10 years	$10,000/level	50	60	50	30	
Fire valves, hoses	20 years	$2.00/ft.	72	110	98	116	
Sump pumps, filters	10 years	$15,000/level	75	90	75	45	
		Subtotal	$ 512	$ 598	$ 528	$ 506	$ 2,144
Rock Dust							
Surface Storage Tank	40 years	$23,000	23	23	23	23	
Access Borehole, lined	40 years	$50,000	50	50	50	50	
Tank, Compressor, Fluidizer, 1 per producing level	40 years	$27,000	54	54	54	54	
Line, 3" fiberglass	10 years	$6.75/ft.	68	68	68	68	
Auxiliary pods	10 years	$25,000	75	75	75	75	
		Subtotal	$ 270	$ 270	$ 270	$ 270	$ 1,080
Communications							
Voice - phone and radio data sensors, telemetry		$50,000	50	50	50	50	200
			$3,945MM	$4,833MM	$4,603MM	$4,516MM	$17,897MM

TABLE 3-17

SUMMARY OF UNDERGROUND SUPPORT CAPITAL COSTS

(Thousands of Dollars)

	Mine #1	Mine #2	Mine #3	Mine #4	Combined
Underground Utilities					
Ventilation	2,040	2,400	2,400	2,400	9,240
Electrical	1,073	1.515	1,355	1,290	5,233
Water	512	598	528	506	2,144
Rock Dust	270	270	270	270	1,080
Communications	50	50	50	50	200
Subtotal	3,945	4,833	4,603	4,516	17,897
Belt Haulage					
Slope Belts, Drive, Structure	1,440	2,453	2,354	1,913	8,160
Main Belts, etc.	2,942	2,275	2,032	1,658	8,907
Submain Belts, etc.	248	1,920	1,758	2,560	6,486
Transfers, Raises, etc.	480	600	600	300	1,980
Subtotal	5,110	7,248	6,753	6,431	25,542
Supply and Personnel Transport					
Locomotives - Diesel	1,255	1,485	1,485	1,485	5,710
Locomotives - Battery	850	850	850	850	3,400
Supply Cars, Lowboys	889	1,241	1,241	1,241	4,612
Scoops, duster, ballast cars, track-mounted bolter, track cleaner	670	670	670	670	2,680
Personnel Elevator	300	300	300	300	1,200
Track	2,520	4,320	4,824	4,320	15,984
Subtotal	6,484	8,866	9,370	8,866	33,586
MINE TOTALS	15,539	20,947	20,726	19,813	77,025

TABLE I-12

SUMMARY OF UNDERGROUND SUPPORT CAPITAL COSTS

(Thousands of Dollars)

	Mine #1	Mine #2	Mine #3	Mine #4	Combined
Underground Utilities					
Ventilation	2,040	2,500	2,400	2,400	9,240
Electrical	1,073	1,315	1,355	1,290	5,233
Water	512	598	328	506	2,184
Rock Dust	220	210	270	270	1,080
Communications	80	50	50	50	200
Subtotal	3,925	4,653	4,603	4,516	17,897
Belt Haulage					
Slope Belts, Drive, Structure	1,440	2,453	2,324	1,913	8,150
Main Belts, etc.	2,942	2,275	2,032	1,658	8,907
Submain Belts, etc.	248	1,320	1,258	2,560	6,486
Transfers, Rakes, etc.	480	600	600	300	1,980
Subtotal	5,110	7,248	6,253	6,431	25,542
Supply and Personnel Transport					
Locomotives - Diesel	1,255	1,885	1,885	1,485	5,710
Locomotives - Battery	850	850	850	850	3,400
Supply Cars, Lowboys	889	1,241	1,241	1,241	4,612
Scoop, duster, ballast cars, track-mounted boiler, track cleaner	670	670	670	870	2,880
Personnel Elevator	300	300	300	300	1,200
Track	2,520	4,320	4,824	3,320	15,984
Subtotal	6,484	9,265	9,270	8,866	33,886
MINE TOTALS	15,519	20,947	20,726	19,813	77,815

Surface facilities at each of the four mine portals will be similar; a typical layout is as shown in Figure 3-11. The major components are as follows:

- Combined Administration/Changeroom/Warehouse/Shop building - Production, safety, maintenance and engineering personnel are provided offices in one portion of the building. Wage earner changerooms are provided for both men and women employees and an additional changeroom is available for line foremen.

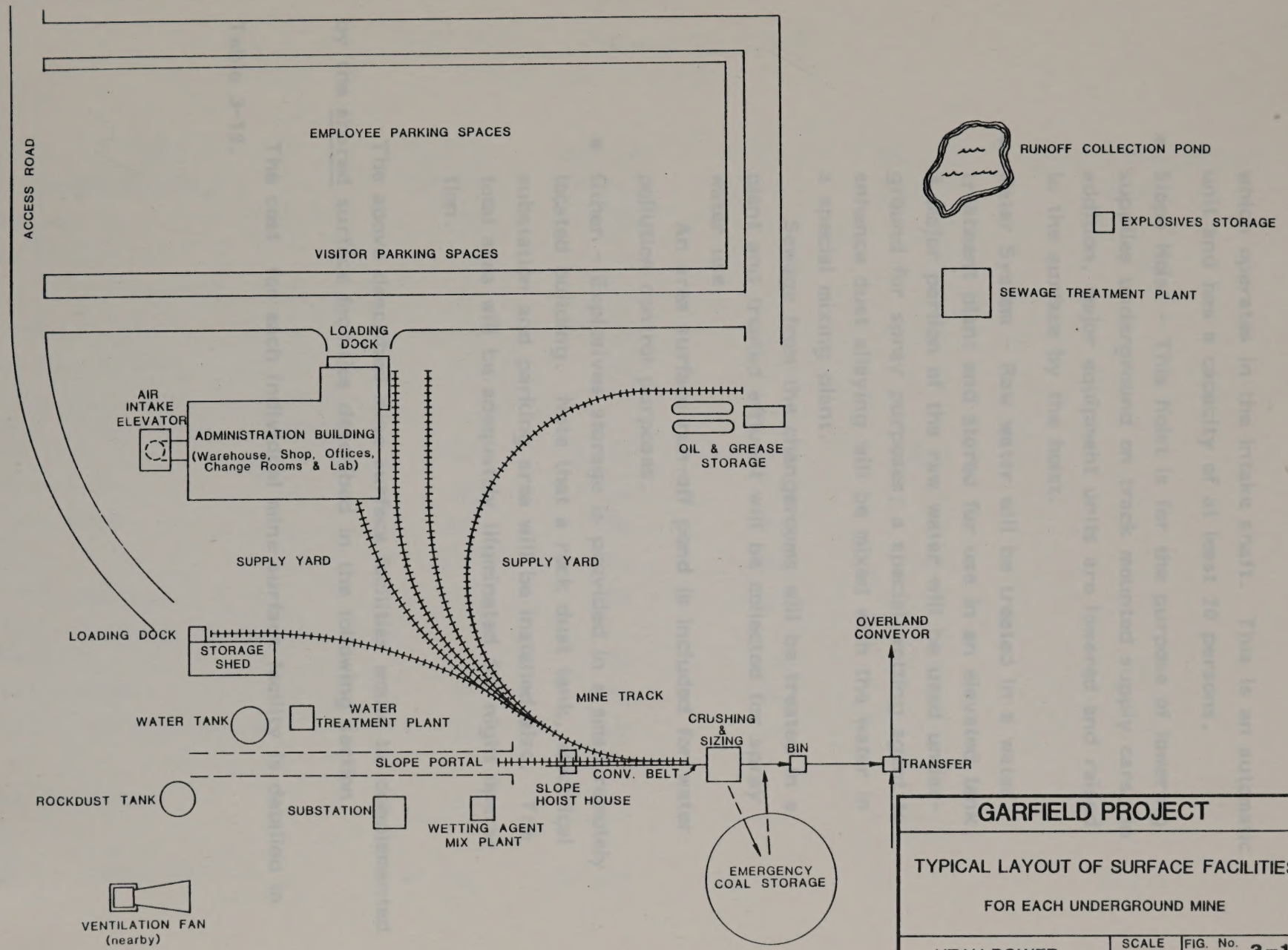
  A large warehouse area is provided for small parts and supplies and other materials requiring weather protection. The warehouse personnel control all incoming materials and machinery and disbursements to the mine operation, and in addition, direct the operation of the supply yard.

  Work conducted in the shop consists of preparing new equipment for use, short term repairs on mining machinery, surface mobile equipment maintenance and repair, and support services for the coal preparation system, mine fans, hoists, etc.

  A quality control laboratory will be included in this building.

- Supply Yard - Bulky materials and machinery are stored in the supply yard. Note that oil and grease storage is provided for large items such as belting, cement, rock dust, etc. requiring weather protection. Also included is mine track which provides for making up supply trips, hauling underground machinery into and out of the shop, etc. A small diesel powered track jitney is utilized for this service.

- Air intake shaft/elevator - Connected to the administration building by an enclosed ramp is the main personnel elevator



GARFIELD PROJECT

TYPICAL LAYOUT OF SURFACE FACILITIES

FOR EACH UNDERGROUND MINE

UTAH POWER & LIGHT CO.	SCALE NONE	FIG. No. 3-11
		DATE AUGUST, 1979

which operates in the intake shaft. This is an automatic unit and has a capacity of at least 20 persons.

- Slope Hoist - This hoist is for the purpose of lowering supplies underground on track mounted supply cars; in addition, major equipment units are lowered and raised to the surface by the hoist.

- Water System - Raw water will be treated in a water treatment plant and stored for use in an elevated tank. A major portion of the raw water will be used underground for spray purposes; a special wetting agent to enhance dust allaying will be mixed with the water in a special mixing plant.

  Sewage from the changerooms will be treated in a plant and treated effluent will be collected for spray water use.

  An area surface run-off pond is included for water pollution control purposes.

- Other - Explosives storage is provided in a small remotely located building. Note that a rock dust tank, electrical substation and parking area will be installed also. The total area will be adequately illuminated for night operation.

The above described mine surface facilities would be complemented by the shared surface facilities described in the following section.

The cost for each individual mine surface facility is detailed in Table 3-18.

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- Slope Hoist - This hoist is for the purpose of lowering supplies underground on track mounted supply cars; in addition, major equipment units are lowered and raised to the surface by the hoist.

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The above described mine surface facilities would be complemented by the shared surface facilities described in the following section.

The cost for each individual mine surface facility is detailed in Table 3-12.

TABLE 3-18

MINE SURFACE FACILITIES COSTS

(Thousands of Dollars)

Item	Mine #1	Mine #2	Mine #3	Mine #4	Central Office/Shop
Administration Building	$2,183	$2,183	$2,183	$2,183	$2,272
Parking Lot	90	90	90	90	33
Supply Yard	4	4	4	4	180
Storage Shed & Explosive Storage	50	50	50	50	
Oil Storage Tanks	20	20	20	20	
Surface Mine Track	78	78	78	78	
Water Treatment Plant, Water Tank, Sewage Treatment Plant, Wetting Agent Plant	973	1,034	1,034	1,059	
Collection Pond	25	25	25	25	
Rock Dust Tank	100	100	100	100	
Substation	125	150	150	150	
Mobile Equipment	350	350	350	350	1,344
Site Prep. (Clearing, Grubbing, Grading)	275	275	275	275	432
Area Illumination	35	35	35	35	30
Surface Electric Distribution	75	75	75	75	70
Shop					1,829
Degreasing Station					80
MINE TOTALS	$4,383	$4,469	$4,469	$4,494	$6,270

Total = $24,085

Common Surface Facilities 3.3

Layout of surface facilities is illustrated in Figure 3-12 and described in the following narrative.

Access and Supply System 3.31

The main access road for all four mine areas will connect to State Route 54 at a point just west of Escalante. There will be approximately 23 miles of road which will generally follow the alignment of an existing county-maintained road that runs southward from Escalante, up Alvey Wash, and through the lease area. A nominal 100-foot wide right-of-way will be required for the access road. Costs for access roads are shown in Table 3-19.

As shown in Figure 3-13 the proposed access road will be 30 feet wide with eight foot shoulders and a drainage ditch on both sides as required. The 30-foot roadway will be paved with a bituminous surface on top of a crushed stone base. This road will provide good all-weather access to all four mine locations, and also keep fugitive dust levels well within air quality regulations. Construction of the access road will consist montly of minor grading of the existing ground, with only a few moderate cut or fill sections required to keep grades to a minimum. Road grades will range from 0 to 6 percent, and curves will range up to 15 degrees.

Alternate access is considered mandatory because of expected traffic levels and flood potential in Alvey Wash. A graded road with gravel surfacing will be located in Right Hand Collet Canyon. This road connects to the main access road about one mile north of the rail loadout plant in the southern part of the lease area. The road would follow the canyon eastward or downgrade for about 16 miles, where it will connect to the gravel road which runs from Escalante to Hole in Rock. This intersection point is located approximately 18.5 miles southeast of Escalante. Occasional road grading and dust control watering will be conducted on an as-required basis. Another improved, but not necessarily paved road is anticipated to be built along a route parallel to the railroad to Glen Canyon City.

3.3 Common Surface Facilities

Layout of surface facilities is illustrated in Figure 3-12 and described in the following narrative.

3.31 Access and Supply System

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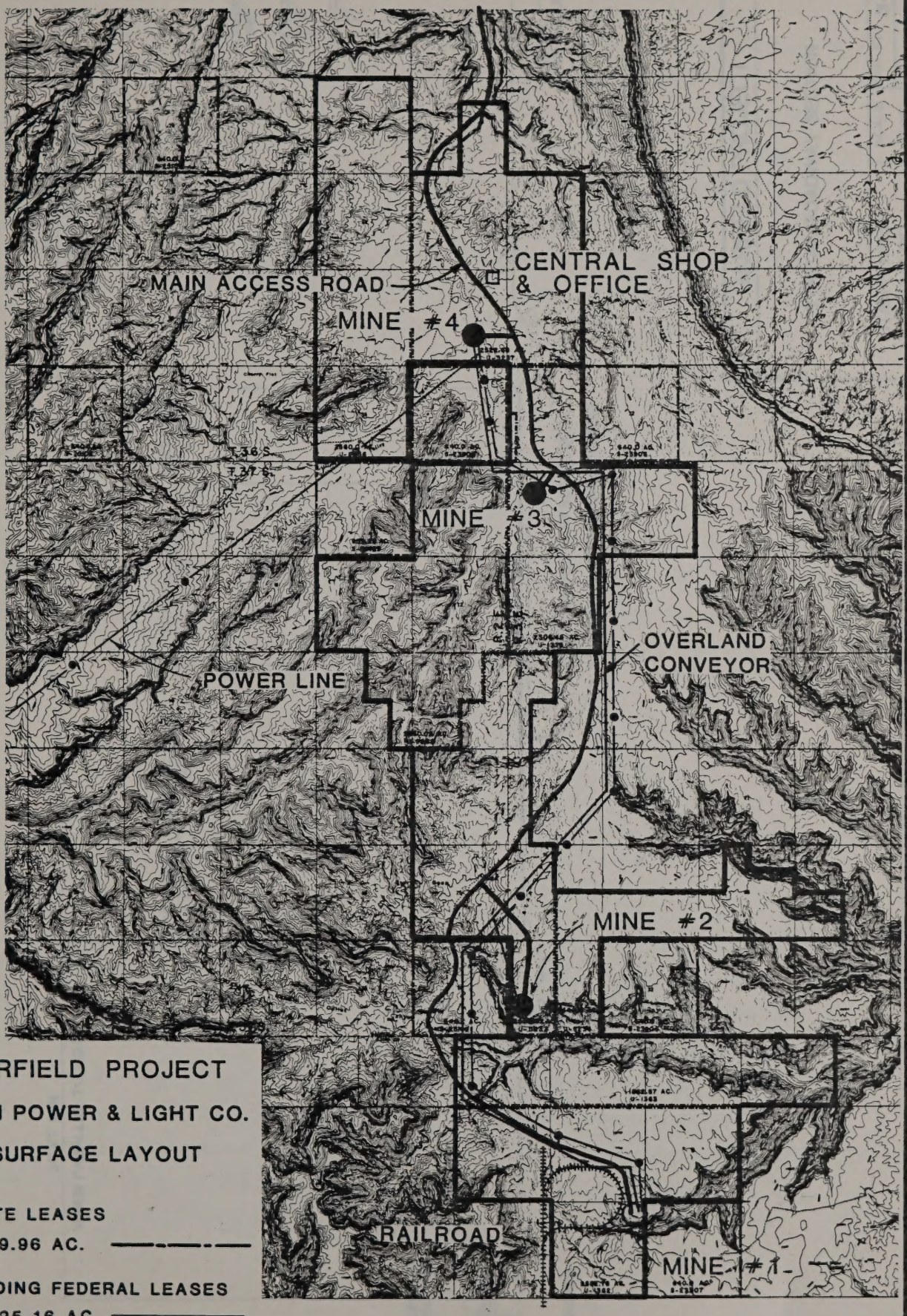


GARFIELD PROJECT

UTAH POWER & LIGHT CO.

SURFACE LAYOUT

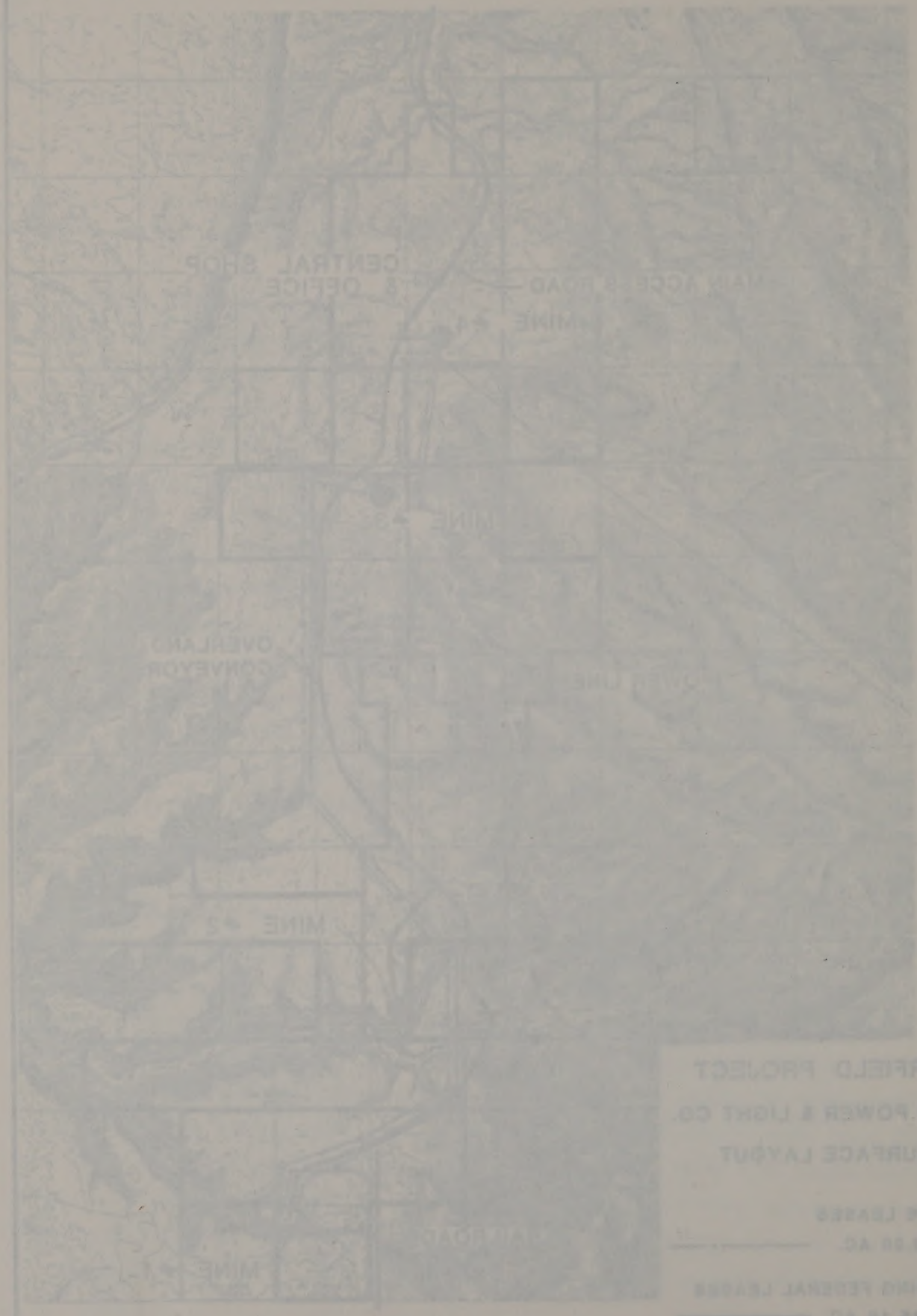
STATE LEASES

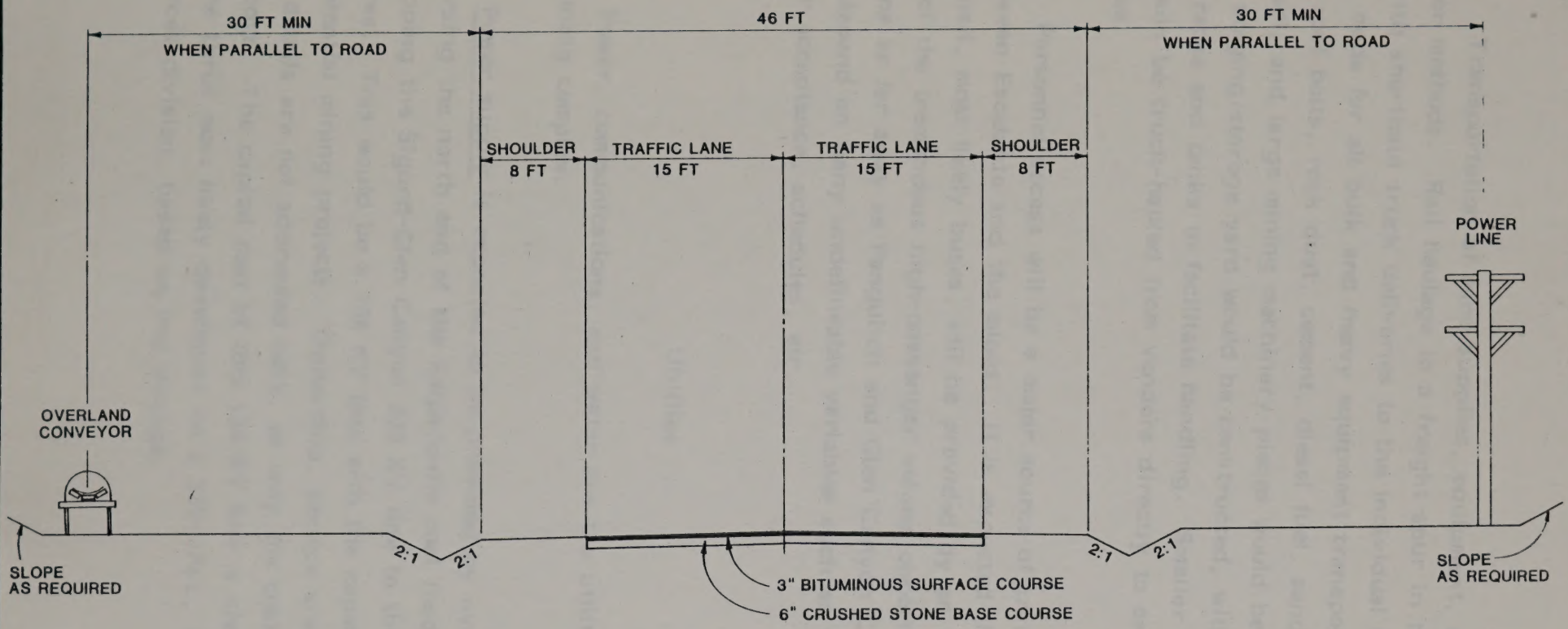
5,119.96 AC. ———-———-———

PENDING FEDERAL LEASES

18,325.16 AC. ———————

FIGURE 3-12

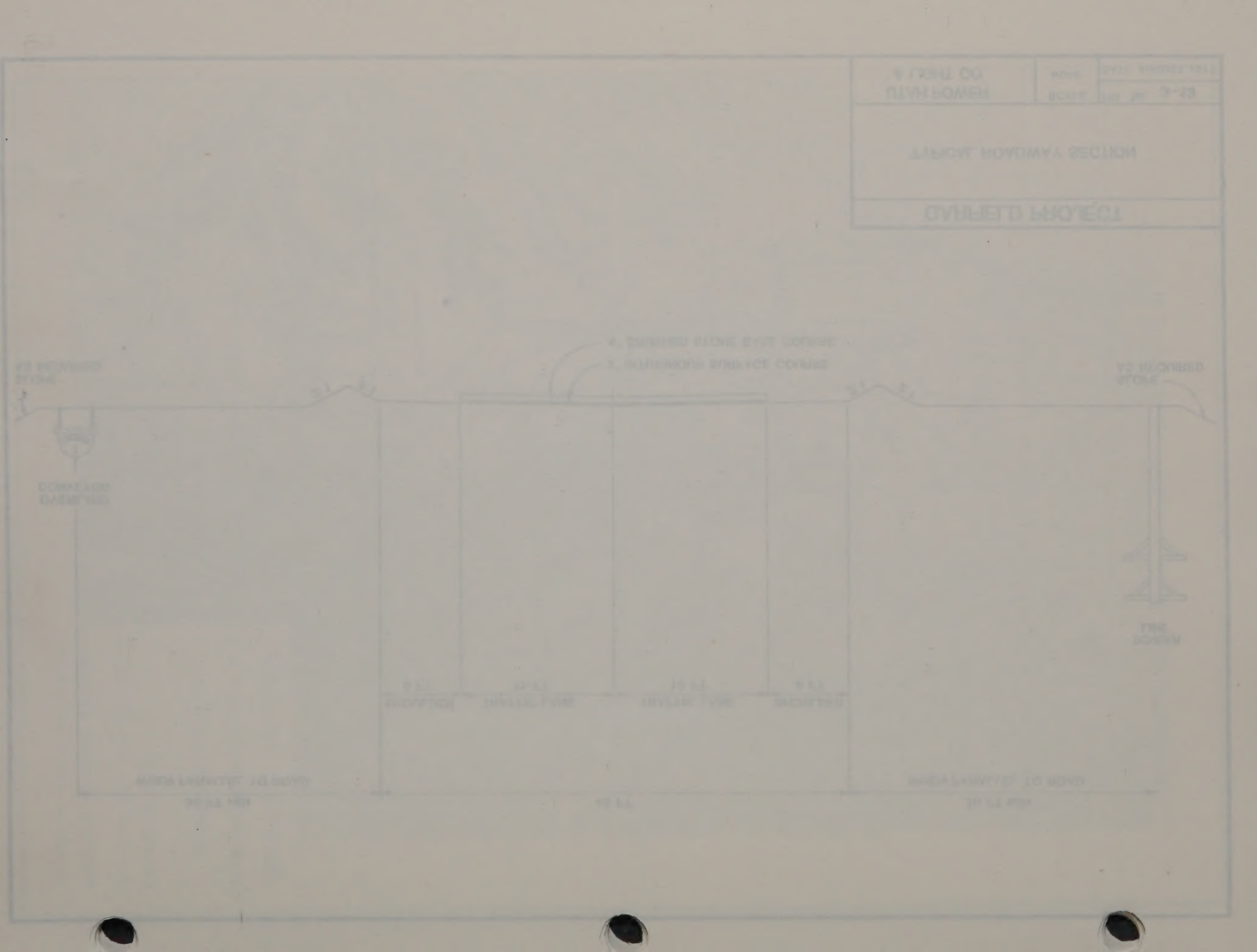




GARFIELD PROJECT

TYPICAL ROADWAY SECTION

UTAH POWER & LIGHT CO.	SCALE NONE	FIG. No. 3-13
		DATE AUGUST, 1979



Transportation of mine supplies, equipment, and parts will be by two major methods. Rail haulage to a freight spur in the loadout area, combined with shorthaul truck deliveries to the individual mines, will be the planned mode for all bulk and heavy equipment transport. Such items as timber, roof bolts, rock dust, cement, diesel fuel, sand and gravel, structural steel, and large mining machinery pieces would be transported by rail. A large staging/storage yard would be constructed, with loading ramps, storage racks and tanks to facilitate handling. Smaller and lighter supply items would be truck-hauled from vendors directly to each mine yard or warehouse.

Personnel access will be a major source of traffic on the paved road between Escalante and the mines. It is expected that some form of mass transit, most likely buses, will be provided by an outside organization because of the tremendous high-passenger volume opportunity. Bussing workers from towns as far away as Panguitch and Glen Canyon City should be feasible, but will depend on many undefineable variables such as future fuel costs, passenger acceptance, schedules, etc.

3.32 Utilities

Power, communications, and water are the utility services required by this mining complex.

Power supply is expected to be provided by overhead high-voltage line traversing the north end of the Kaiparowits coal field from a major substation tapping the Sigurd-Glen Canyon 230 KV line to the UP&L Garfield Deposit area. This would be a 138 KV line with the capacity to service several potential mining projects. Ownership, service areas, and other operating details are not addressed here, as only the costs are relevant to this report. The capital cost of this 138 KV line is shared in this analysis between the three most likely developers on a 50%-UP&L, 30%-El Paso, and 20%-Resources division, based on line mileage.

A large transformer is required to reduce the regional grid voltage of 230 KV to the 115 KV projected for the Kaiparowits line. This expenditure (nearly $5 MM) is shared equally by the three developers in this analysis.

Power distribution on the Garfield Project property is expected to continue at 115 KV to four major load centers (#4 Mine-Central Shop, #3 Mine, #2 Mine and #1 Mine-Loadout) where substations would be located for optimum distribution to the surface and underground facilities, most likely at 13.8 KV and 4.16 KV. Power for the overland conveyors will be provided from each mine site.

Capital costs for power supply are included in Table 3-19.

Communications for this mining project are estimated to require an entirely new telephone exchange with microwave transmission to the regional grid as well as buried cable connections with the existing telephone system in Escalante. Although not planned in detail, approximately 100 phone sets with around 30 to 40 outside lines are expected to be required.

Additionally, an inter-mine radio communications system with base stations at six sites and about ten mobile units is projected. Costs for the communication system are shown in Table 3-19.

Computer services for accounting, engineering, and maintenance-warehouse functions are anticipated to be housed at the Central Office, with terminal access located in each department at each mine. Costs for a small mainframe, disc memory, and twenty terminals are included in Table 3-19.

Water requirements for all the mines will total 380,000 gallons per day:

	Consumption (gpd)			
Use	Mine 1	Mines 2, 3 and 4	Central	Total
Spray Water	52,000	78,000		286,000
Potable Water	18,000	23,000	7,000	94,000
Total	70,000	101,000 per mine	7,000	380,000

A well can be drilled to a depth of about 4000' to the Navajo Sandstone at each mine site, to provide the water needed at each mine, at a cost of approximately $400,000, including drilling, casing, pump, and controls. Water from each well would be piped into a storage tank large enough to supply water to a mine for two days (140,000 to 200,000 gallons). The water can be routed to a small water treatment plant from the storage tank to prepare potable water for employee use or to add a dust retardant/wetting agent to the spray water for dust control in the mine and the coal handling system. Sewage effluent will be properly treated and returned to the mine for spray water use. The potable water for the central building will be treated and stored at Mine #4, less than one mile distant. The mine water storage tanks are interconnected between two mines for redundancy of supply. At each mine, there will be an emergency storage tank to provide enough water for fire fighting requirements.

The costs of the water storage and handling system are listed in Table 3-19.

TABLE 3-19

PROJECT ACCESS AND UTILITIES COSTS

(Thousands of Dollars)

ACCESS ROADS		
I. Main Access Road		
A) Clearing and Grubbing	$ 188	
B) Excavation & Grading	3,875	
C) Road Materials	2,280	
D) Drainage Structures	322	
	$6,665	
II. Alternate Access Road		
A) Clearing Grubbing	$ 35	
B) Excavation & Grading	440	
	$ 475	
	$7,140	
POWER SUPPLY		
I. Primary Substation (230.138 KV)	$3,470 x 1/3 share =	$1,160
II. Line (138 KV)	2,900 x 1/2 share =	1,450
Communications	400 x 1/2 share =	200
III. Mine Site (Transformers & Distrib.)	3,230 x 100% =	3,230
		$6,040
COMMUNICATIONS		
I. Telephone		
A) Phones & Installation	$ 88	
B) Lines & Poles	212	
	$ 300	
II. Radio		
A) Mobile Units	$ 22	
B) Base Stations	13	
	$ 35	
COMPUTER SERVICES		
Mainframe	$ 300	
Disc Memory	100	
Terminals (20)	90	
High speed printer (1)	70	
Low speed printer (6)	36	
	$ 596	
WATER		
4 Water storage tanks (60-80,000 gal)	$ 204	
4 Fire Water Tanks (100-120,000 gal)	500	
4 Potable Water Storage Tanks (20,000 gal)	80	
4 Water Treatment Plants	110	
4 Sewage Treatment Plants	320	
4 Wetting Agent Mixing Plant	160	
Piping between mines 1+2, 3+4	1,073	
4 Wells (4000 ft. incl. pump, casing, controls)	1,600	
	$4,047	
TOTAL ACCESS & UTILITIES	$18,158	

These facilities will be located approximately 8.5 miles south of Escalante and 1.1 miles north of #4 mine on the east side of the paved access road. The central office building will be a pre-engineered steel structure enclosed with metal siding and situated on a concrete foundation. This building will be 200 feet long, 100 feet wide, with a 12-foot eave height. The structural system used throughout the building will be a post and beam type. This office will house management, administrative, engineering, business, and other related services personnel. Also located in this building will be a computerized systems monitoring facility capable of monitoring all mining acitivities, a training room, a large and small conference room, a kitchen and lunch room, and all other required facilities.

The central shop will also be a pre-engineered steel structure enclosed with metal siding and situated on a concrete foundation. It will be 160 feet long, 100 feet wide, with a 24-foot eave height, and a post and beam type structural system. This shop will be utilized for major equipment repairs and overhauls, which can not be accomplished at the individual mine shops.

There will be five repair shops which will occupy approximately 60 percent of the building. It will contain a mining equipment shop, where all continuous miners, longwall units, roof bolter, etc. will be repaired. Additionally, there will be a mobile equipment shop, a hydraulic repair shop, an electrical shop, and a machine shop. The warehouse, change room, and office will occupy the remainder of the building. The offices will be occupied by supervisory personnel for the shops and warehouse. A lunch room and training room will also be located in the building.

A large storage yard, enclosed by chain link fence, will be located near the shop. This yard will provide storage space for bulky items that are not affected by the weather. A degreasing station will also be located in the supply yard.

A 200 car employee parking lot, located across from the office and shop, will provide adequate parking facilities for all employees working in the office and shop.

Costs for this Central facility are included in Table 3-18, located in Section 3.26.

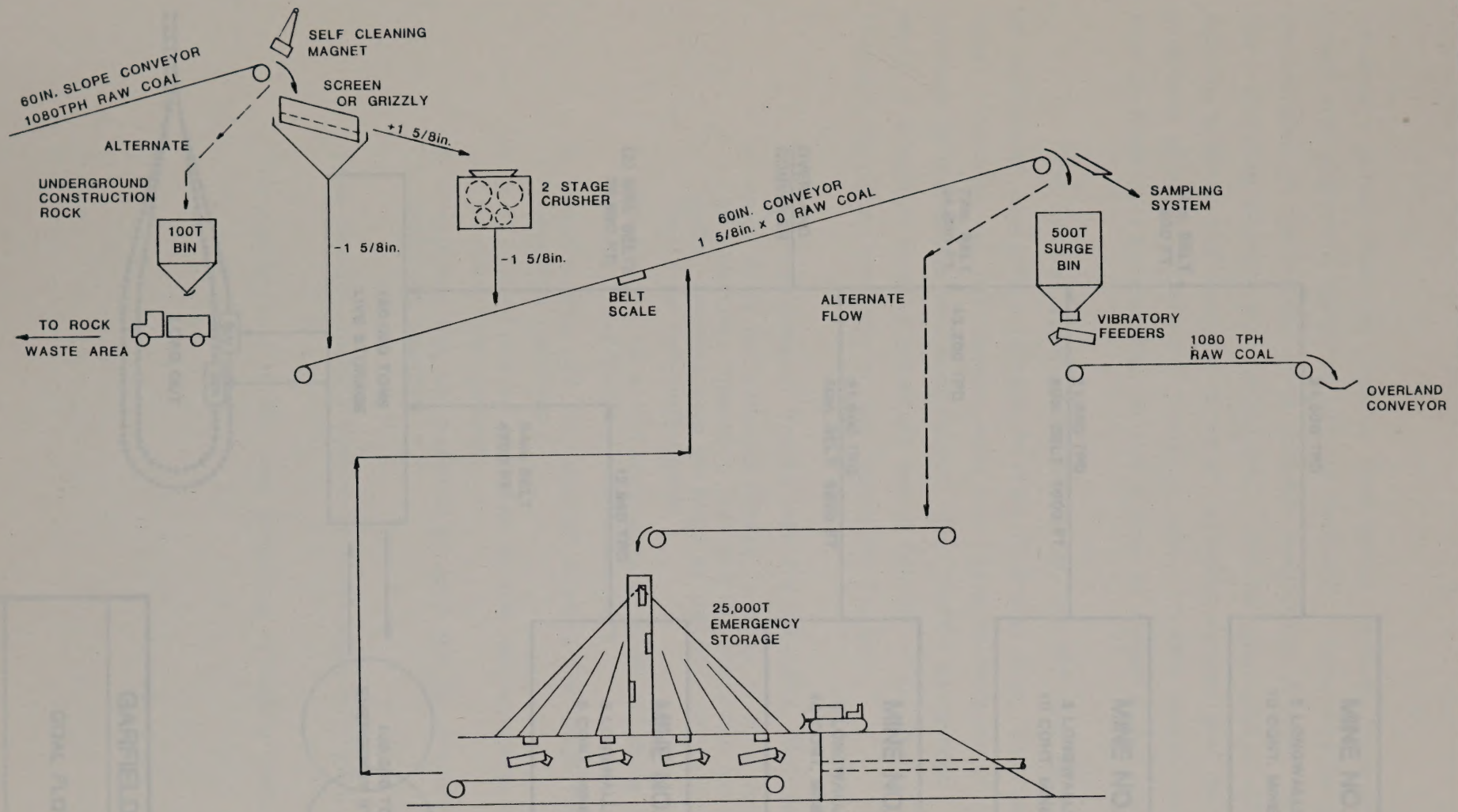
3.34 Coal Handling System

At each mine, coal will be brought to the surface on a slopebelt conveyor and be discharged into a coal preparation system as shown in Figure 3-14. The purpose of this system are as follows:

- reduce the ROM coal to a top size of 1 5/8 inches
- provide flow-rate smoothing through use of a surge bin in preparation for feeding the overland conveyor
- provide alternate emergency storage in case of overland conveyor or other downstream delays
- provide tramp iron cleaning and underground construction rock handling, and,
- provide for mine production monitoring through a belt scale and a coal sampling system.

As shown in Figure 3-15, the total overland conveyor system will be approximately 14 miles long and consist of at least 19 major segments. Note that two parallel conveyors are used between Mine 2 and the loadout storage area in order to increase the system reliability through redundancy. Also, note that the Mine 1 product is conveyed independently to the loadout storage. (Appendix 3-2).

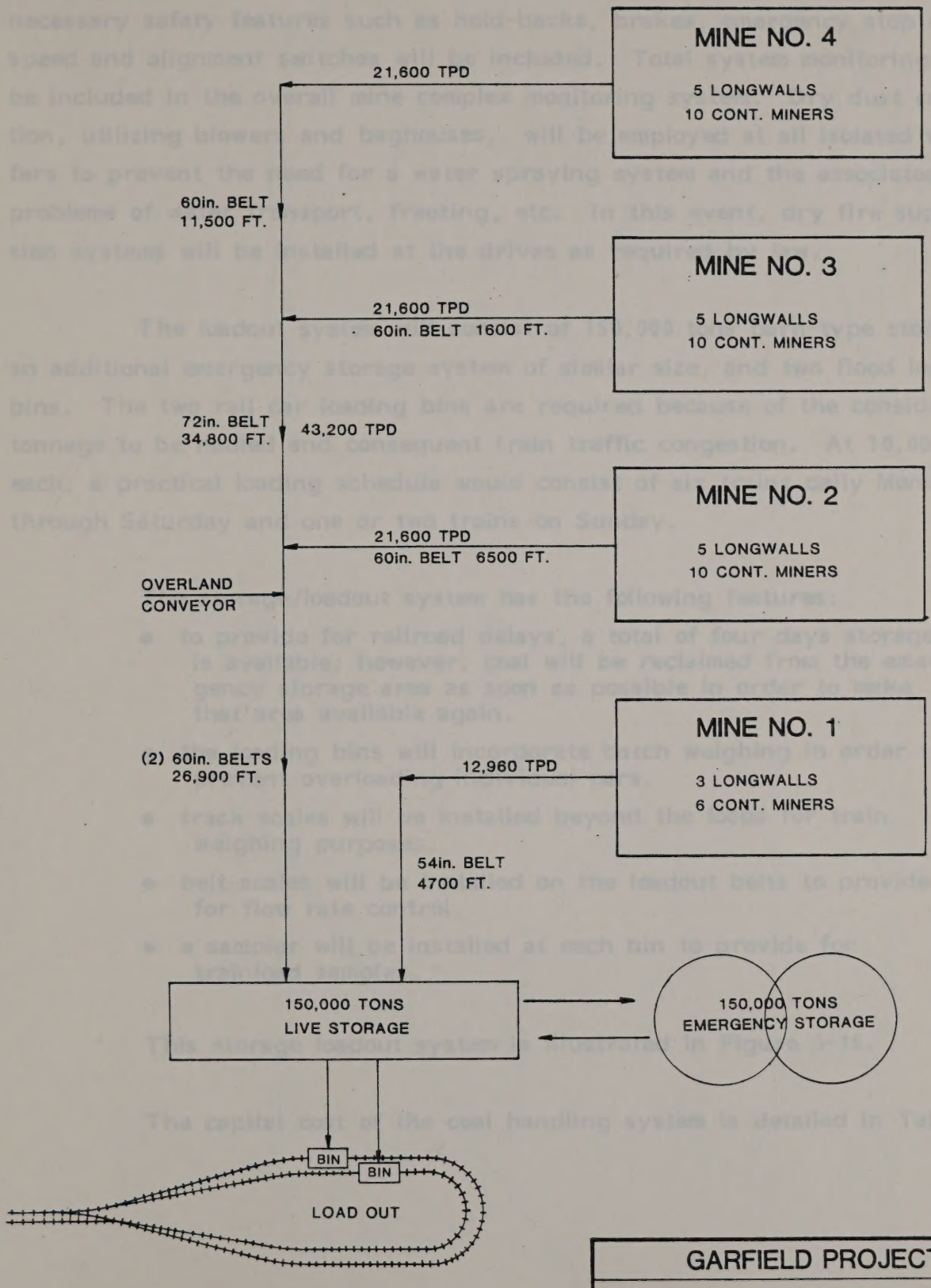
The overland conveyor route appears quite satisfactory as far as grades, etc. are concerned; in fact, much of the route is on relatively flat plateau-like areas and few draws must be traversed requiring rigid trusses. In addition, access for construction purposes is good due to the route generally paralleling an existing dirt road. The conveyor will be hooded and



GARFIELD PROJECT

TYPICAL MINE-SITE
COAL PREPARATION AND HANDLING

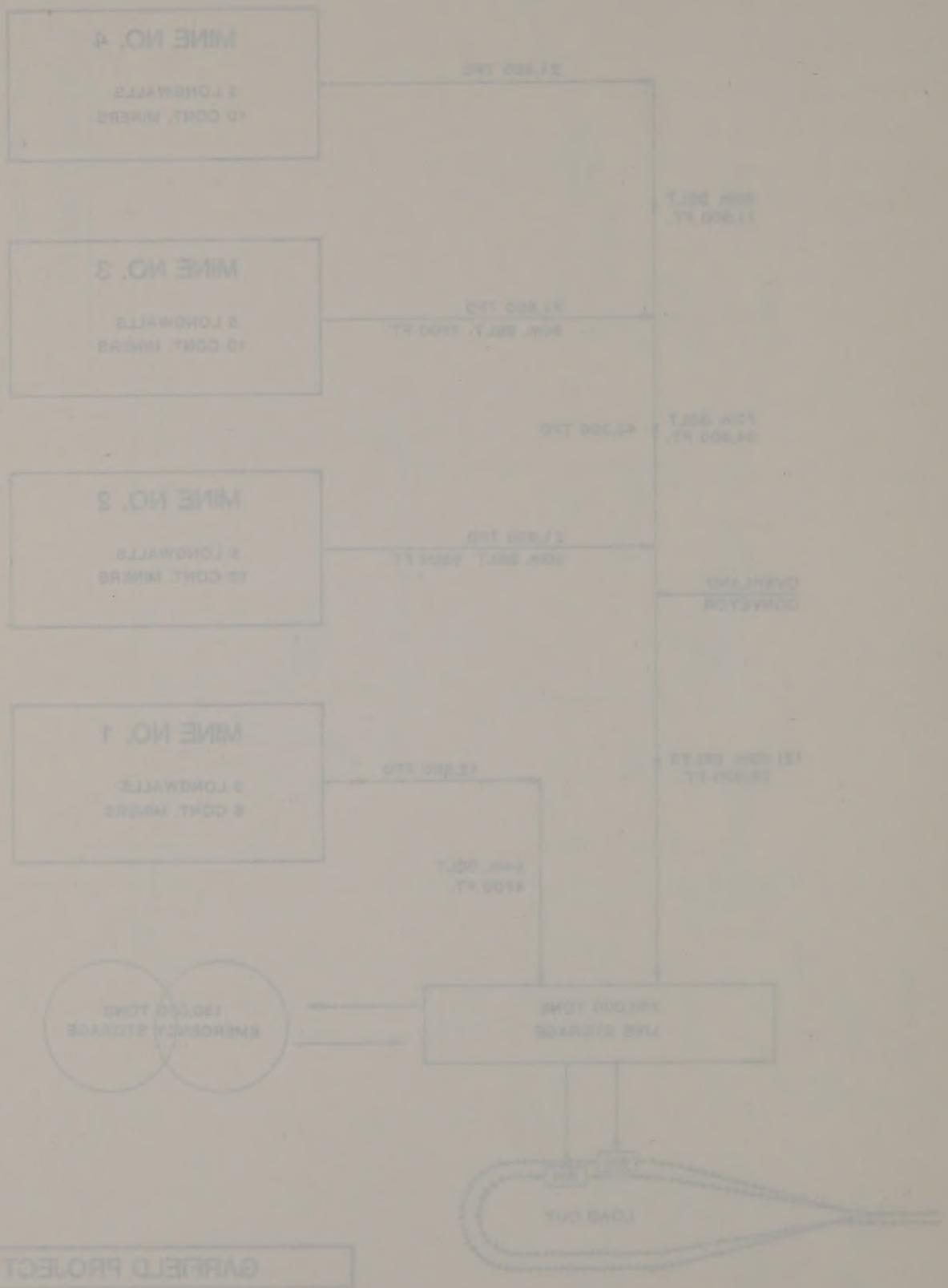
UTAH POWER & LIGHT CO.	SCALE NONE	FIG. No. 3-14
		DATE AUGUST, 1979



GARFIELD PROJECT

COAL FLOW SCHEME

UTAH POWER & LIGHT CO.	SCALE NONE	FIG. No. 3-15
		DATE AUGUST, 1979



necessary safety features such as hold-backs, brakes, emergency stop cords, speed and alignment switches will be included. Total system monitoring will be included in the overall mine complex monitoring system. Dry dust collection, utilizing blowers and baghouses, will be employed at all isolated transfers to prevent the need for a water spraying system and the associated problems of water transport, freezing, etc. In this event, dry fire suppression systems will be installed at the drives as required by law.

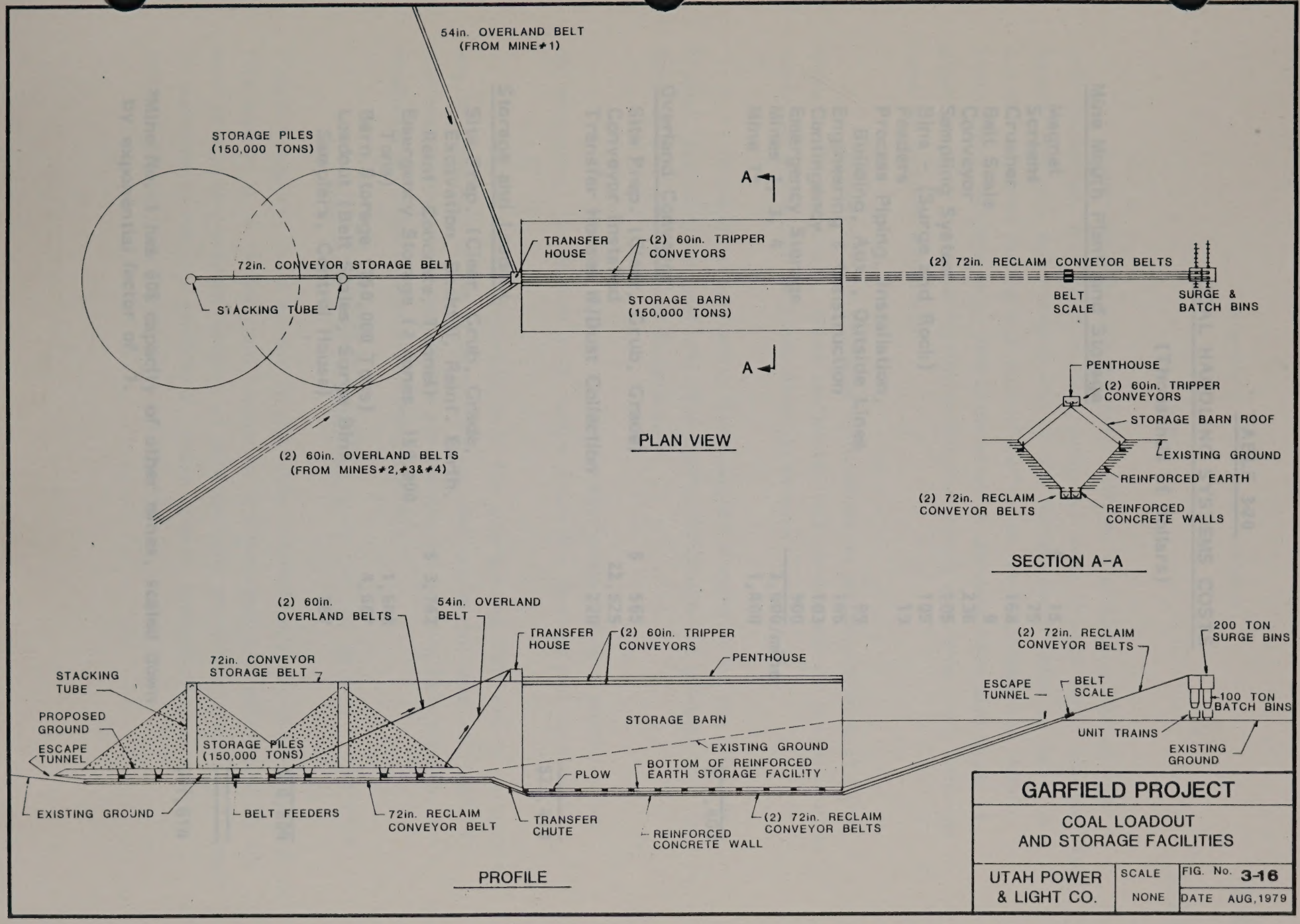
The loadout system will consist of 150,000 tons barn-type storage, an additional emergency storage system of similar size, and two flood loading bins. The two rail car loading bins are required because of the considerable tonnage to be hauled and consequent train traffic congestion. At 10,000 tons each, a practical loading schedule would consist of six trains daily Monday through Saturday and one or two trains on Sunday.

The storage/loadout system has the following features:

- to provide for railroad delays, a total of four days storage is available; however, coal will be reclaimed from the emergency storage area as soon as possible in order to make that area available again.
- the loading bins will incorporate batch weighing in order to prevent overloading individual cars.
- track scales will be installed beyond the loops for train weighing purposes.
- belt scales will be installed on the loadout belts to provide for flow rate control.
- a sampler will be installed at each bin to provide for trainload samples.

This storage loadout system is illustrated in Figure 3-16.

The capital cost of the coal handling system is detailed in Table 3-20.



GARFIELD PROJECT

COAL LOADOUT AND STORAGE FACILITIES

UTAH POWER & LIGHT CO.	SCALE NONE	FIG. No. 3-16
		DATE AUG, 1979

TABLE 3-20

COAL HANDLING SYSTEMS COSTS

(Thousands of Dollars)

Mine Mouth Plant and Storage		
Magnet	$ 15	
Screens	75	
Crusher	168	
Belt Scale	9	
Conveyor	236	
Sampling System	105	
Bins - (Surge and Rock)	105	
Feeders	19	
Process Piping, Installation, Building, Aux., Outside Lines	99	
Engineering & Construction	166	
Contingency	103	
Emergency Storage	900	
Mines 2, 3, 4	2,000/mine	
Mine 1*	1,400	
		$ 7,400
Overland Conveyor		
Site Prep. (Clear, Grub, Grade)	$ 565	
Conveyor Installed	22,625	
Transfer Houses W/Dust Collection	220	
		$23,410
Storage and Loadout		
Site Prep. (Clear, Grub, Grade, Excavation, Backfill, Reinf. Earth, Reinf. Concrete, Tunnel)	$ 3,762	
Emergency Storage (2 cones, 150,000 Tons)	1,648	
Barn Storage (150,000 Tons)	4,607	
Loadout (Belt Scales, Surge Bins, Samplers, Control House)	787	
		$10,804
		$41,614

*Mine No. 1 has 60% capacity of other mines, scaled down by exponential factor of .7.

Manning 3.41

Labor requirements for each shift on a producing face section were developed, multiplied by the number of faces in each mine, and tabulated. Support workers underground and on the surface were listed by job title and the required numbers of each classification were estimated in detail. Supervisory, engineering, and administrative workers were also listed in detail for each mine, correlating production levels and crew sizes with the need for salaried personnel.

Crew manning estimates were based on actual practices at UP&L mines in Central Utah. Other personnel lists were based on previous mine supervision and planning-operating experience.

Table 3-21 summarizes the expected workforce for the Garfield Project, including General and Mine Staffs (both salaried groups) and the Surface and Underground Wageearners.

The precise salaries or wage classification of each employee are not estimated. In Section 3.42, Labor Costs are derived and explained.

This estimated workforce of 431 salaried and 2407 hourly personnel may be compared to the projected annual coal production level of 18 MMTPY and the scheduled 230 workshifts per person per year to find the overall productivity projected for this mining complex:

$$\frac{18{,}000{,}000 \text{ tons / year}}{2838 \text{ workers} \times 230 \text{ shifts/year}} = 27.56 \text{ tons / man-shift}$$

This productivity is in excess of published national industry averages, because this project is to be designed, equipped and managed to produce coal by the most productive methods known, in good geologic conditions (thick, flat seams) in moderate to low gas conditions. Many modern longwall mines commonly achieve productivity levles even greater than projected here.

Operating Cost Determination

3.41 Manning

Labor requirements for each shift on a producing face section were developed, multiplied by the number of faces in each mine, and tabulated. Support workers underground and on the surface were listed by job title and the required numbers of each classification were estimated in detail. Supervisory, engineering, and administrative workers were also listed in detail for each mine, correlating production levels and crew sizes with the need for salaried personnel.

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The precise salaries or wage classification of each employee are not estimated. In Section 3.42, Labor Costs are derived and explained.

This estimated workforce of 431 salaried and 2807 hourly personnel may be compared to the projected annual coal production level of 13 MMTPY and the scheduled 230 workshifts per person per year to find the overall productivity projected for this mining complex:

$$\frac{13{,}000{,}000 \text{ tons / year}}{2238 \text{ workers} \times 230 \text{ shifts/year}} = 27.55 \text{ tons / man-shift}$$

This productivity is in excess of published national industry averages, because this project is to be designed, equipped and managed to produce coal by the most productive methods known, in good geologic conditions (thick, flat seams) in moderate to low gas conditions. Many modern longwall mines commonly achieve productivity levels even greater than projected here.

TABLE 3-21

SALARIED EMPLOYEES

GENERAL STAFF

Supervision General Manager (1), General Superintendents (2-Production & Maintenance), Clerical (4)	7
Administrative Superintendent (1), Purchasing and Payroll Staff (5), Clerical (8)	14
Labor Relations Superintendent (1), Training and Safety Staff (10), Security Staff (9), Clerical (2)	22
Technical Superintendent (1), Chief Engineers (4, by discipline), Staff Engineers (8), Drafting and Clerical (6)	19
Subtotal General Staff	62

MINE STAFFS

	Mine #1	Mine #2	Mine #3	Mine #4	Loadout	Central
Mine Superintendent	1	1	1	1	1	–
General Foreman	1	1	1	1	1	–
Shift Foreman	7	7	7	7	3	–
Face Bosses - Section & Weekend Special	33	53	53	53	–	–
Maintenance & Supply Foreman	9	9	9	9	–	–
Maintenance Superindendent	1	1	1	1	–	1
Maintenance Foreman	3	3	3	3	1	3
Clerical	5	5	5	5	–	5
Plant Superintendent	–	–	–	–	–	1
General Foreman	–	–	–	–	–	3
Mine Engineer	1	1	1	1	–	–
Staff	4	4	4	4	–	2
Safety Engineer & Staff	3	3	3	3	–	1
Mine Administrator & Staff	3	3	3	3	–	3
Loadout Operations	–	–	–	–	–	–
	71	91	91	91	6	19

Subtotal Mine Staffs	369
Total Salaried Employees	431

TABLE 3-21a

WAGE EARNERS

	Mine #1	Mine #2	Mine #3	Mine #4	Central Shop	Coal Handling System
Face Production						
Longwall	123	205	205	205	–	–
Continuous Miner	138	230	230	230	–	–
General Underground						
Electricians, Mechanics & Welders	15	17	17	17	–	–
Construction & Equipment Movers	71	85	85	85	–	–
Delivery & Dispatching, Housekeeping	37	45	45	45	–	–
Surface						
Coal Preparation	10	10	10	10	–	21
Electrician, Welder, & Mechanics	13	13	13	13	24	9
Construction Inspection, Housekeeping	5	5	5	5	8	6
Supply Yard	8	9	9	9	5	1
Overhaul & Rebuild Mechanics	–	–	–	–	34	–
Belt & Cable Repair	–	–	–	–	12	–
Drivers, Road Maintenance, & Snow Removal	–	–	–	–	10	–
	420	619	619	619	93	37

Total Wage Earners 2,407

Labor Cost

Labor cost is composed of three components:

- Payroll
  - salaries of supervisory, technical and clerical workers
  - hourly wages of all skilled and unskilled mine workers
- Overhead on Payroll
  - UMWA Welfare Fund (per man-hour payment)
  - Social Security Tax
  - workmens compensation insurance
  - fringe benefits for both hourly and salaried employees
- Overhead on Production
  - UMWA Welfare Fund (per ton payment

Data provided by UP&L for the labor cost at their mines on the Wasatch Plateau are used for this analysis. These mines are UMWA mines, and so for worst-case analysis, the Kaiparowits mines are assumed to be similarly organized. The weighted average daily UMWA base rate is $74.27 in UP&L's Central Utah mines. A variety of benefits cause a substantial amount of overhead to be added to this amount to get the total daily labor cost. As a percentage of the base rate, direct benefits are as follows:

Vacation Pay	6 %
Holidays	4.5%
Sick Leave	2 %
Clothing Allowance	1 %
	13.5%

Indirect benefits are as follows:

UMWA Pension Funds	42.5%
Federal Black Lung	6.5%
Insurance	9.5%
State Workmen's Comp	4.5%
Social Security (FICA) and Unemployment Ins.	7.5%
	70.5%

The above percentages were provided by UP&L. Thus, the total daily cost per hourly worker is $136.66.

At the Wasatch Plateau mines, UP&L and/or its mine operating contractor had 324 people on salary in the month of April 1979, with a total payroll of $661,359, an average salary of $2,041 per month or $24,500 per year. Presuming 260 working days per year per employee yields an average daily salary expense of $94.20 per day, which includes the costs of holidays and vacations. Benefits, both direct and indirect are estimated at 45% of direct salary and would include health insurance, unemployment insurance, workmen's compensation, social security, and pension and profit sharing plans. The total daily cost per salaried employee is $136.59.

Labor costs, then, can be calculated as follows:

431 salaried employees x $136.59/day x 260 days/year	=	$15.306 MM/yr
2407 hourly employees x $136.66/day x 230 days/year	=	75.656 MM/yr
		$90.962 MM/yr

On a per ton basis, the cost of labor is:

$$\frac{\$90.962 \text{ MM/yr}}{18 \text{ MMTPY}} = \$5.05/\text{ton}$$

Because of the worst case assumption involving a United Mine Workers labor force, a UMWA Pension Fund contribution of $1.385 per ton must be added to the above labor cost, yielding a total cost of labor of $6.44 per ton.

Supply and Power Costs 3.43

Supplies, materials, power, and fuel are considered to be direct operating costs supporting a variety of unit operations. These direct operating costs are composed of several major components:

- expendable materials
- repair parts and materials
- lubricants
- electrical power and fuel

In this analysis the following items are included as expendable materials: miner and longwall bits, roof bolts, timbers, rock dust, brattice, blocks, track and conveyor supplies, electrical cable and cable repairs, tires, tubes, greases, and oils, small tools, and repair parts. In a study prepared for the Electric Power Research Institute[1], it was found that there is a consistent relationship between labor cost and supplies. Since labor costs are easily quantified, and since this study provides a proven relationship, the following assumptions to develop supply costs estimates are made:

Mine Size

The authors of the report concluded that, based on data from fifty mines, when the sum of straight time and overtime wages and salaries and all overhead on hourly labor per ton was plotted against materials and supplies per ton, there was a linear relationship. Not only were continuous mining systems considered, but also conventional and longwall as well, and the linear relationship still held. Mine sizes of 200,000 TPY to 2,500,000 TPY were considered. The technique followed in the EPRI report is followed here.

Not one of the 50 U.S. mines is over 2½ mm tons/yr.

As was reported earlier in this report, the weighted average daily UMWA direct wage rate at UP&L mines on the Wasatch Plateau is $74.27 per day and is considered representative for the Garfield Project. The various overhead items on hourly labor amount to an additional 84% of daily wage. An additional payment of $1.385 per ton produced is required by the 1978 UMWA wage agreement. The weighted average direct salary of supervisory and clerical personnel is $94.20 per day, based on current supervisory-engineering-administrative payroll costs in the UP&L deep mines in Utah.

Supply costs may be estimated using the EPRI model formula which considers labor costs (less benefits for salaried employees):

Salaried employees: 431 x $94.20/day x 260 days/yr	= $10.556 MM/yr
Hourly employees: 2407 x $136.66/day x 230 days/yr	= 75.656
	$86.212 MM/yr
- 18 MMTPY	= 4.79/ton
+ UMWA fund	1.38
Labor rate for model imput	= $ 6.17/ton

[1]NUS Corporation (1977) Coal Mining Cost Models - Underground Mines: EPRI Report EA 437.

Inserting this modified labor cost into the EPRI model yields this supply cost:

supply cost = 1.387 + 0.924 ($6.17 = $4.31/ton

Power costs for the typical underground mine may also be similarly derived following the EPRI model:

power cost = .0158 + .0681 ($6.17) = $0.44/ton

Because the EPRI-NUS coal cost model analyzes only underground coal production with minimal surface coal handling, the cost of power for the large overland belt system for the Garfield Project is considered an additional increment. The following table presents the estimated annual power costs for all surface facilities extraordinary to the model:

Cost of Power Not Included in EPRI Model

1) Central Shop and Office -
1000 HP x .746 KWH/HP x 18 Hr/Day x 230 Days/Yr x $0.03/KWH = $ 93,000

2) Coal Loadout and Storage -
1500 HP x .746 KWH/HP x 20 Hr/Day x 300 Days/Yr x $0.03/KWH = $ 201,000

3) Overland Conveyors -
22,500 HP x .746 KWH/HP x 24 Hr/Day x 230 Days/Yr x $0.03/KWH = $2,780,000

TOTAL = $3,074,000

Annual power cost/Ton = $3,074,000/Yr - 18,000,000 Tons/Year = $0.17/Ton

Other Costs

In addition to the above labor, supply and power costs, various taxes and royalties may be considered as operating cost components.

Federal Black Lung (MSHA) and Federal Reclamation (OSM) funds are supported by a direct tax on production. These taxes are $0.65 per ton.

Inserting this modified labor cost into the EPRI model yields this supply cost:

supply cost = 1.387 + 0.924 ($6.17 = $9.31/ton

Power costs for the typical underground mine may also be similarly derived following the EPRI model:

power cost = .0158 + .0681 ($6.17) = $0.44/ton

Because the EPRI-NUS coal cost model analyzes only underground coal production with minimal surface coal handling, the cost of power for the large overland belt system for the Garfield Project is considered an additional increment. The following table presents the estimated annual power costs for all surface facilities extraordinary to the model:

Cost of Power Not Included in EPRI Model

1) Central Shop and Office -
1000 HP x .746 KWH/HP x 16 Hr/Day x 230 Days/Yr x $0.03/KWH = $ 33,000

2) Coal Loadout and Storage -
1500 HP x .746 KWH/HP x 20 Hr/Day x 300 Days/Yr x $0.03/KWH = $ 201,000

3) Overland Conveyors -
22,830 HP x .746 KWH/HP x 24 Hr/Day x 230 Days/Yr x $0.03/KWH = $2,780,000

TOTAL = $3,014,000

Annual power cost/Ton = $3,014,000/Yr ÷ 15,000,000 Tons/Year = $0.17/Ton

3.44

Other Costs

In addition to the above labor, supply and power costs, various taxes and royalties may be considered as operating cost components.

Federal Black Lung (MSHA) and Federal Reclamation (OSM) funds are supported by a direct tax on production. These taxes are $0.65 per ton.

The use tax is applied to tangible personal property purchased for use or consumption in Utah. It is presumed that the 4% tax is included as part of the purchase price of each item and is not included as an expensable operating cost, even though it legally could have been.

Property (ad valorem) tax is based on a capitalization of income approach for existing mines. For new mines, the valuation for tax purposes is the cash value of improvements and tangible personal property as established by the local assessor. In this analysis, it is assumed that one-half the initial mine investment is equivalent to the average free market value of the project, that this value is assessed at 33% and that amount taxed at 70 mills. This dollar amount, presumed to occur in each of the 38 production years, is multiplied by 38 and then divided by total, life-of-mine production to get a cost per tone, which is $0.44/ton.

Liability and disaster insurance is estimated to be $0.0035 per dollar of invested capital, which for this analysis was the initial investment. The cost per ton is $0.08.

Lessor royalty is assumed to be 8%, payable on both the federal and state leases. The percentage is applied to the FOB mine price. Overriding royalty, payable to previous lessees of these properties totals $.07 per ton.

Design engineering of the entire four-mine complex will include many features intended primarily to protect the natural environment during construction and mining. Regulatory requirements imposed by the Office of Surface Mining, the Bureau of Land Management, as well as other Federal and State agencies will be met as a minimum.

Minimizing surface disturbance, erosion and sedimentation, water effluents, and fugitive dust emissions are usually the principal concerns of these agencies, so specific treatment of these areas was included in the cost estimates for various surface facilities. To the extent possible, it is intended that the mining complex be a closed system, with coal dust and waste water emissions added to the coal product.

Road dust will be minimized by paving, auto exhaust emissions will be reduced by the encouragement of an employee oriented bus system, coal dust generated at transfer points and in the loadout areas will be controlled by both water sprays and vacuum enclosures with "baghouse" dust collectors, the in-line coal storage will be partially underground and covered to prevent wind-generated coal dust. Topsoil salvage and replacement, regrading and seeding, with spray irrigation in the early years as necessary, will be done on all construction-affected lands not required for mining activity.

Wastewater from the office and shop facilities will be treated for reuse in the mine dust spray system. Excess groundwater entering the mine, if any, will also be treated as necessary for use in the dust suppression spray system. NPDES permits will be obtained just in case there are unexpected large amounts of groundwater inflow.

Coal and rock wastes will be stowed underground where allowable by safety regulations, but a major portion will require disposal in controlled fills near the mine sites, with adequate ditching, ponds, and topsoil material for land restoration included in the plan.

Design engineering of the entire four-mine complex will include many features intended primarily to protect the natural environment during construction and mining. Regulatory requirements imposed by the Office of Surface Mining, the Bureau of Land Management, as well as other Federal and State agencies will be met as a minimum.

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Coal and rock wastes will be stowed underground where allowable by safety regulations, but a major portion will require disposal in controlled fills near the mine sites, with adequate ditching, ponds, and topsoil material for land restoration included in the plan.

Immediately following cessation of mining or as any facility is phased out of production, reclamation will be initiated. Preliminary estimates are that only a few acres will be directly affected by surface installations. Office and shop/maintenance buildings will be removed with their foundations and their sites will be regraded to approximate original contour. Reseeding to establish a productive plant community will follow. The overland conveyor system will be dismantled and its foundations and any access roads will be regraded and revegetated. If animal crossing provisions have been made along the conveyor pathway, these will be removed and the area restored to as near original condition as possible. In all cases with surface facility removal the best available methods to prevent erosion and enhance return to a near natural state will be used. Coal loadout facilities will probably be salvaged and any storage areas cleared of coal and debris before they are rehabilitated. In each case all due regard will be given to adequate treatment or removal of any mine related materials before final reclamation. Of special concern will be the removal of potentially toxic or undesirable materials i.e. coal refuse, construction debris, and hydrocarbon or chemical contaminants around shop, warehouse, and storage areas.

The underground access areas, ventilation shafts, slope portals, and exhaust fan openings, will be permanently sealed by prescribed requirements in force at the time of closure and the areas surrounding them returned to original contour and revegetated. All reclamation will be designed to meet legal requirements and to return the area to as near natural conditions as possible. It is expected that the railroad will continue its useful life beyond that of the specific mine project and its eventual reclamation will be accomplished with like concern for the environment.

Sealing and reclamation costs are believed to be relatively minor, especially in view of their very late timing in the project life. Salvage value of facilities is likely to be greater than final reclamation costs, so neither are considered in the economic analysis.

The capital and operating costs of environmental controls and reclamation are not presented in a separate table because they have been included in each productive and supportive aspect of the entire mine plan. These provisions are integral to the design and proper function of the entire mining system and are therefore not isolated.

3.6 Coal Product Quality

In order to estimate the quality of the coal product, the basic quality data provided by coal cores has been modified to provide for mining dilution (rock from roof and floor) and adjustments of the moisture content:

- Dilution -- Some rock from the top and bottom will be mined in varying amounts dependent on seam thickness, seam variability, type of mining machine, etc. Considering the many variables, four inches of dilutant have been added to the basic core quality data as a representative dilution. The dry dilutant material is considered to have a specific gravity of 2.5, with 80% ash, 1000 BTU/lb, and no sulfur content. In general, four inches of this mateial represents approximately seven weight percent dilution.

- Moisture --It is anticipated that the total moisture of the "As Shipped" product will be approximately 11.5 percent. Considerations used in arriving at this value are:
  1. The everage "As Received" (in the laboratory) moisture content of the UP&L cores is approximately 15.0% (see Table 2-13, columns B and C). General knowledge of these coals suggests this is quite high and that it is due to wet drilling and determination of the total moisture of unusually wet samples in plastic bags. Discussions with UP&L personnel verify this assumption.
  2. Moisture analyses on cores taken on a nearby lease from the same Christensen Zone seams show much lower moistures, averaging about eight percent. The handling technique of these cores is known, and consisted of rinsing the cores to remove drilling mud, allowing a short period for drainage, then bagging in plastic containers. Equilibrium moisture analyses run on these samples are of the sam general value, confirming the validity of the core handling method and of the assumption of eight percent moisture .
  3. Calculation of the MAF-BTU)moisture- and ash-free) of the UP&L and nearby lease analyses described above further supports the similarity of the seams and hence suggests the moistures would be similar.
  4. Two higher zones of coal will also be mined in the UP&L mines. The "As Received" moisture contents of cores cut in these zones are practically the same as the moistures determined in the Christensen Zone.

Coal Product Quality

Therefore, it is assumed that these cores were handled in the same manner and that the equilibrium moistures would also be similar to those of the nearby coal lease.

Based on the above, it has been assumed that a moisture value about midpoint between the UP&L data and the data from the adjacent property is more realistic for the in-place coal seams of the Garfield Deposit. Thus, the in-place moisture content is estimated to be 10.0%.

The addition of dust spray water at numerous points throughout the mine and coal handling system is estimated to add about 1.5% moisture to the product coal, yielding a run-of-mine moisture content of 11.5%.

Table 3-22 lists the anticipated run-of-mine quality for seams that are to be mined in the proposed mining plan, based on a derived average moisture content of 10.0%. These average analyses are weighted by seam tonnage to be recovered. This is a blended average, with no consideration of seam production scheduling.

TABLE 3-22

PROJECTED RUN-OF-MINE QUALITY

Mine	Moisture	Ash	Sulfur	BTU
1	11.5%	13.5%	0.78%	10,190
2	11.5	12.6	0.78	10,230
3	11.5	12.2	0.73	10,270
4	11.5	12.4	0.73	10,200
Blend	11.5%	12.6%	0.75%	10,230

3.7 Summary of Mine Development Costs

Within each of the previous sections of this report, capital cost tabulations were presented by major function:

Underground Development

Table 3-9	Seam Access
Table 3-12	Production Units
Table 3-17	Support Functions

Surface Development

Table 3-18	Mine Buildings
Table 3-19	Access and Utilities
Table 3-20	Coal Handling

These cost tabulations are recapitulated in Table 3-23, Summary of Initial Capital Investment. This table presents capital investment for each mine, for common or shared facilities, and the allocated cost of railroad branch-line construction (which will be discussed in Section 4).

Development of the Garfield Project will occur over a sixteen-year period; the capital expenditures detailed in Section 3 are combined with the development schedule in Table 5-1 and presented in Section 5 - Economic Analysis.

TABLE 3-23

SUMMARY OF INITIAL CAPITAL INVESTMENT

(Millions of Dollars)

	Mine #1	Mine #2	Mine #3	Mine #4	Common	Total	Source
Predevelopment					15.11	$ 15.11	
EIS, drilling, engineering, permit applications.							
Underground Facilities & Equipment							
Seam Access	$ 9.78	$17.39	$16.35	$13.83		57.35	Table 3-9
Production Units	35.61	59.38	59.38	59.38	38.96*	252.71	Table 3-12
Belts, Haulage, Utilities	15.54	20.95	20.72	19.81		77.02	Table 3-17
*spare equipment not assigned to a mine						$387.08	
Surface Facilities & Equipment							
Mine Buildings	4.38	4.47	4.47	4.49	6.27	24.08	Table 3-18
Access & Utilities					18.16	18.16	Table 3-19
Coal Handling System	1.40	2.00	2.00	2.00	34.21	41.61	Table 3-20
						$ 83.85	
Mine Totals	$66.71	$104.19	$102.92	$99.51	112.71	$486.04	
Railroad (Big Sage Branch, Prorated share of capital costs)					79.52	79.52	
Total Initial Capital Investment						$565.56	

4 RAILROAD

4.0 RAIL TRANSPORTATION

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4.0 RAIL TRANSPORTATION

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4. Rail Transportation

Southern Utah Coal Transportation Alternatives 4.1

Development of the coal resources of southern Utah, both the underground-mineable coal seams in the Kaiparowits coal field and the surface-mineable coal seams in the Alton coal field, will require the construction of a major new transportation system. Because of the location of the coal fields and the single product to be shipped, a transport system can be selected and optimized for this entire area.

Potential Coal Shipment Levels 4.11

Coal production estimates for the coal fields of southern Utah if diligent development were to occur for all leases and PRLA's would be on the order of 120 MMYPY. Production estimates developed in a 1977 study prepared by the Union Pacific Railroad (UPRR), assuming a more realistic development schedule for all potential producers, indicated that about 50 MMTPY would be produced for the coal fields (Appendix 4-1). The Southern Utah Coal EIS (1979) estimated a mid-level production rate of 29 MMTPY and a high-level rate of 46 MMTPY. These estimates would indicate that a potential for producing about 40 MMT of coal annually from the coal fields of southern Utah exists. For purposes of this study, proposed development plans evaluated on a site specific basis in the southern Utah coal EIS combined with the UP&L development plan assure an annual production rate of 40 MMT with the coal originating from UP&L, El Paso, Resources Co., and Utah International. All producers would utilize the same transportation system.

4. Rail Transportation

Southern Utah Coal Transportation Alternatives

Development of the coal resources of southern Utah, both the underground-minable coal seams in the Kaiparowits coal field and the surface-minable coal seams in the Alton coal field, will require the construction of a major new transportation system. Because of the location of the coal fields and the single product to be shipped, a transport system can be selected and optimized for this entire area.

Potential Coal Shipment Levels

4.12 Mine-Mouth Generation

UP&L, as well as other companies, has advocated mine-mouth generation for the coals of the Kaiparowits Plateau because of the obvious economic advantages of transporting energy of transmission line rather than by rail or slurry pipeline. However, at the present time and for purposes of this study, a scenario whereby coal will be mined and shipped to consumers for use in coal-fired power plants at locations far from the coal fields is followed.

4.13 Slurry Pipeline

Coal slurry pipelines have been studied and evaluated, in a preliminary sense, for transporting coal from the coal fields of southern Utah to markets in central Utah, southwestern Utah, and Nevada. The lack of flexibility in allowing the transport of escalating and variable quantities of coal and the difficulties in dewatering slurried coal makes this alternative unattractive for the persent. Another significant deterrant to the utilization of a slurry pipeline is the high premium placed on water in the semi-arid climate of southern Utah due to the lack of availability of surface water.

4.14 Railroad Transportation

The escalating nature and variability of annual production from the coal fields of southern Utah as well as the potential markets for these coals indicated, at an early date, that the presence of a railroad would satisfy the evolving demands of potential coal producers and suppliers. The closest railroads are located to the west and north of the coal fields and early studies concentrated on identifying potential railroad routes from the coal deposits to existing rail haulage systems (Figure 4-1). Later studies, particularly one

Society For Applied Spectroscopy

Rocky Mountain Section

Denver, Colorado

April 29, 1980

Mr. Alan James Hurd
c/o Department of Physics
University of Colorado at Boulder
Boulder, Colorado 80309

Dear Mr. Hurd:

It gives me great pleasure to inform you that you have been selected to receive the Outstanding Student Award in recognition of your exceptional contributions in the field of spectroscopy. This award is presented annually by the Rocky Mountain Section of the Society for Applied Spectroscopy. The presentation will be made at a Tuesday evening banquet during the Twenty-second Annual Rocky Mountain Conference on Analytical Chemistry and Applied Spectroscopy to be held at the Denver Convention Center Complex, August 11-14th.

As Student Award recipient, we invite you to present a short paper at the Conference that will describe important aspects of your recent research. Your paper will very likely be scheduled on Monday afternoon, August 11th, and will be designated as the "Student Award Address." At your earliest convenience would you please provide me with the title and length of your talk (about 20-25 minutes is generally appropriate).

The Student Award consists of an engraved plaque and a cash award of One Hundred Dollars. In addition you will receive complimentary registration for the August Conference and will be a guest of the Section at the Award Banquet. You may expect to receive a copy of the preliminary program of the Conference within the next 2 or 3 weeks.

Please accept my sincere congratulations on your selection for this singular recognition. We look forward to meeting you and to the opportunity to offer our congratulations in person. Best Wishes as you continue your work at the University of Colorado.

Sincerely yours,

Marvin W. Skougstad
Chairman, Student-Award Committee

cc: Steve Valente, Chairman
Rocky Mountain Section, SAS
Professor W. J. O'Sullivan
Professor Noel A. Clark

Society For Applied Spectroscopy

Rocky Mountain Section

Denver, Colorado

April 19, 1980

Mr. Alan James Burt
c/o Department of Physics
University of Colorado at Boulder
Boulder, Colorado 80309

Dear Mr. Burt:

It gives me great pleasure to inform you that you have been selected to receive the Outstanding Student Award in recognition of your exceptional contributions in the field of spectroscopy. This award is presented annually by the Rocky Mountain Section of the Society for Applied Spectroscopy. The presentation will be made at a Tuesday evening banquet during the Twenty-second Annual Rocky Mountain Conference on Analytical Chemistry and Applied Spectroscopy to be held at the Denver Convention Center, Denver, August 11-14th.

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Mervin V. Skougstad
Chairman, Student-Award Committee

cc: Steve Valaora, Chairman
Rocky Mountain Section, SAS
Professor W. J. O'Sullivan
Professor Neal A. Clark

Society For Applied Spectroscopy

Rocky Mountain Section

Denver, Colorado

April 29, 1980

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c/o Department of Physics
University of Colorado at Boulder
Boulder, Colorado 80309

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Chairman, Student-Award Committee

cc: Steve Valente, Chairman
Rocky Mountain Section, SAS
Professor W. J. O'Sullivan
Professor Noel A. Clark

Society For Applied Spectroscopy

Rocky Mountain Section

Denver, Colorado

April 29, 1980

Mr. Alan James Ford
c/o Department of Physics
University of Colorado at Boulder
Boulder, Colorado 80309

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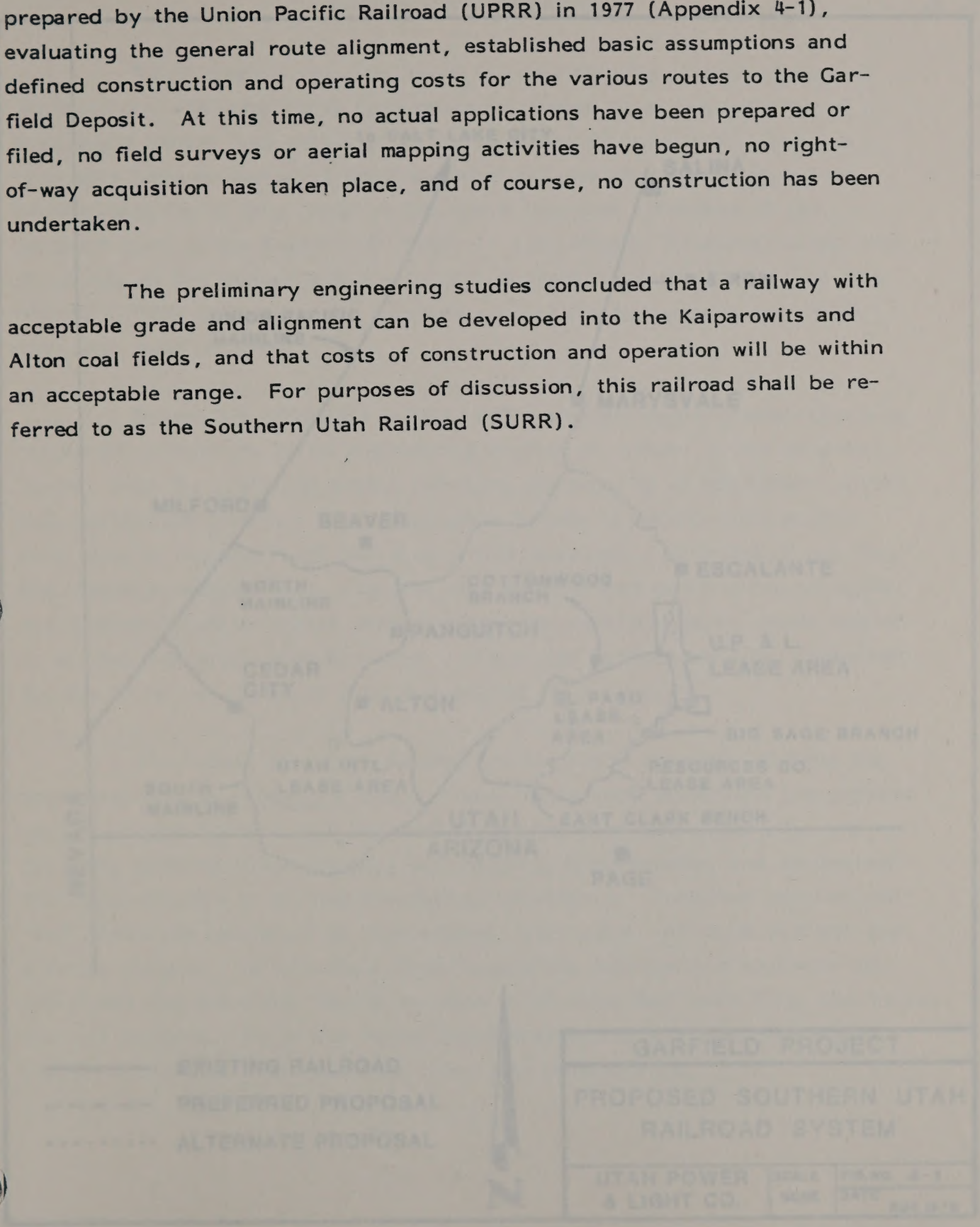
Sincerely yours,

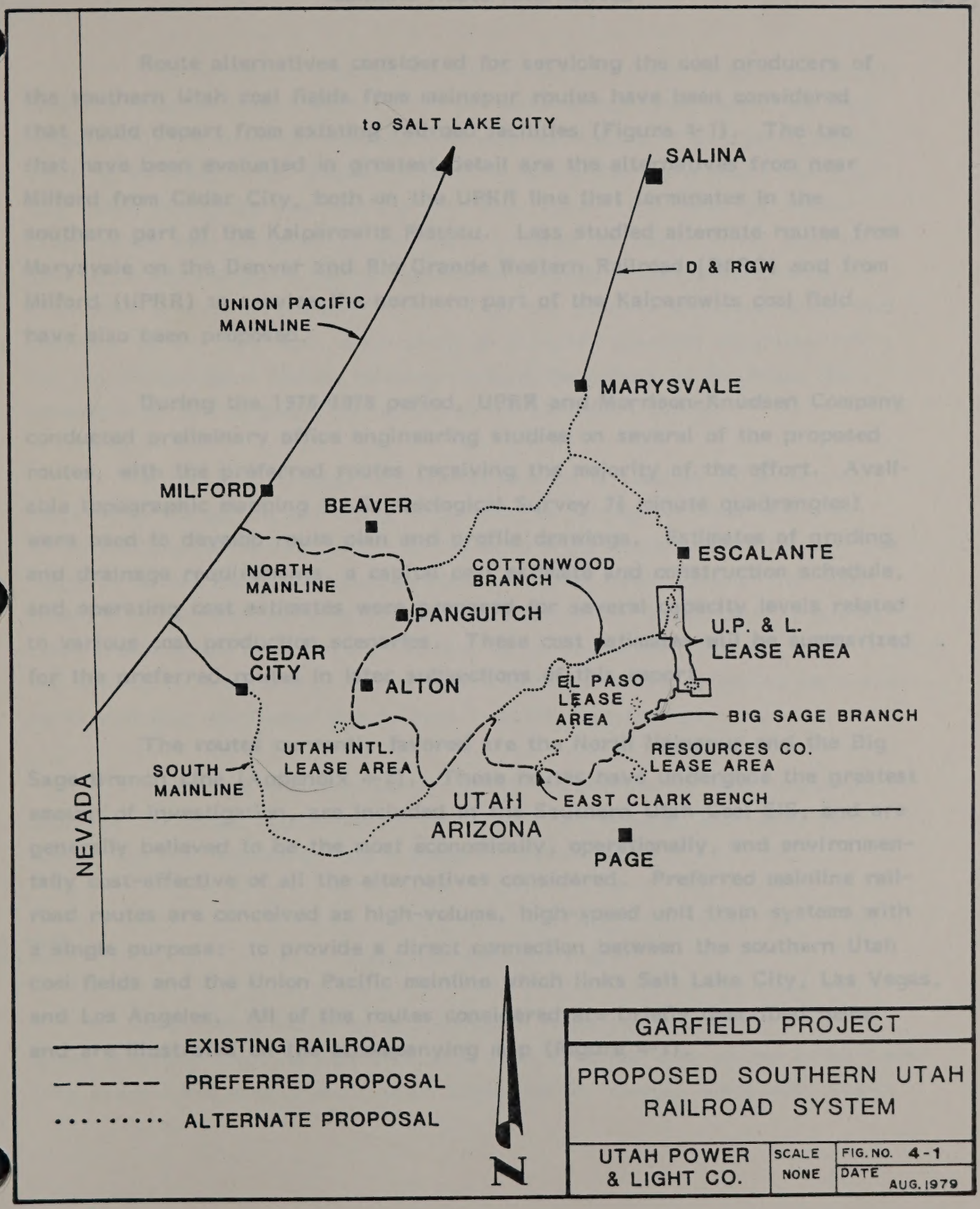
Marvin V. Sheppard
Chairman, Student-Award Committee

cc: Steve Vadasz, Chairman
Rocky Mountain Section, SAS
Professor W. J. O'Sullivan
Professor Noel A. Clark

prepared by the Union Pacific Railroad (UPRR) in 1977 (Appendix 4-1), evaluating the general route alignment, established basic assumptions and defined construction and operating costs for the various routes to the Garfield Deposit. At this time, no actual applications have been prepared or filed, no field surveys or aerial mapping activities have begun, no right-of-way acquisition has taken place, and of course, no construction has been undertaken.

The preliminary engineering studies concluded that a railway with acceptable grade and alignment can be developed into the Kaiparowits and Alton coal fields, and that costs of construction and operation will be within an acceptable range. For purposes of discussion, this railroad shall be referred to as the Southern Utah Railroad (SURR).




to SALT LAKE CITY
SALINA
D & RGW
UNION PACIFIC MAINLINE
MARYSVALE
MILFORD
BEAVER
NORTH MAINLINE
COTTONWOOD BRANCH
ESCALANTE
PANGUITCH
U.P. & L. LEASE AREA
CEDAR CITY
ALTON
EL PASO LEASE AREA
BIG SAGE BRANCH
UTAH INTL. LEASE AREA
RESOURCES CO. LEASE AREA
SOUTH MAINLINE
UTAH
EAST CLARK BENCH
ARIZONA
PAGE
NEVADA
EXISTING RAILROAD
PREFERRED PROPOSAL
ALTERNATE PROPOSAL
N
GARFIELD PROJECT
PROPOSED SOUTHERN UTAH RAILROAD SYSTEM
UTAH POWER & LIGHT CO.
SCALE NONE
FIG. NO. 4-1
DATE AUG. 1979

GARFIELD PROJECT

PROPOSED SOUTHERN UTAH
RAILROAD SYSTEM

UTAH POWER
& LIGHT CO.

SCALE
NONE

DATE

EXISTING RAILROAD

PREFERRED PROPOSAL

ALTERNATE PROPOSAL

Mainline Route Alternatives 4.2

Route alternatives considered for servicing the coal producers of the southern Utah coal fields from mainspur routes have been considered that would depart from existing railroad facilities (Figure 4-1). The two that have been evaluated in greatest detail are the alternatives from near Milford from Cedar City, both on the UPRR line that terminates in the southern part of the Kaiparowits Plateau. Less studied alternate routes from Marysvale on the Denver and Rio Grande Western Railroad (D&RG) and from Milford (UPRR) to service the northern part of the Kaiparowits coal field have also been proposed.

During the 1976-1978 period, UPRR and Morrison-Knudsen Company conducted preliminary office engineering studies on several of the proposed routes, with the preferred routes receiving the majority of the effort. Available topographic mapping (U.S. Geological Survey 7½ minute quadrangles) were used to develop route plan and profile drawings. Estimates of grading and drainage requirements, a capital cost estimate and construction schedule, and operating cost estimates were prepared for several capacity levels related to various coal production scenarios. These cost estimates will be summarized for the preferred routes in later subsections of this report.

The routes currently favored are the North Mainspur and the Big Sage Branch Line (Appendix 4-2). These routes have undergone the greatest amount of investigation, are included in the Southern Utah Coal EIS, and are generally believed to be the most economically, operationally, and environmentally cost-effective of all the alternatives considered. Preferred mainline railroad routes are conceived as high-volume, high-speed unit train systems with a single purpose: to provide a direct connection between the southern Utah coal fields and the Union Pacific mainline which links Salt Lake City, Las Vegas, and Los Angeles. All of the routes considered are briefly described below and are illustrated on the accompanying map (Figure 4-1).

4.21 Northern Mainspur Route

The northern route for the SURR mainline would originate approximately 8 miles south of Milford, Utah on the existing Union Pacific mainline (Figure 4-1). The route would extend in a southeasterly direction passing south of Minersville and Beaver, Utah. It would then turn south through Dog Valley and cross over the rim into Bear Valley, then into the Sevier River Valley near Panguitch. The route crosses the Sevier River then begins an ascending grade past Panguitch and Hatch to the summit north of Alton with a maximum grade of 1.5%. From the summit, the route would run generally to the southeast, descending on a nearly constant 1% grade, skirting the Paunsaugunt Plateau passing through leaseholds in the Alton coal reserve area to Telegraph Flat south to the Vermillion Cliffs.

If the line through Cottonwood Canyon is selected to serve the Kaiparowits Plateau, a gathering yard could be located on Telegraph Flat. If the decision is reached to utilize the more southeasterly route through the Kaiparowits coal field (Big Sage Branch), the line would turn to the north from Telegraph Flat and descend into the Paria River Valley on a 1% grade. Upon crossing the Paria River, the line turns southward along the river through the Cockscomb and ascends to East Clark Bench where a gathering yard would be located and the feeder line to Area 1 or Big Sage commences.

The route would include 11 bridge structures for major roads or drainage crossings, approximately ten 12,500 foot sidings, microwave for communications, signaling, and a weigh-in-motion scale.

4.22 Southern Mainspur Route

Also being examined is a southern route for the main track which would originate approximately five miles west of Cedar City on the UPRR Cedar City Branch and extend south down Cedar Valley towards Hurricane, thence

south and east towards Virgin, Utah (Figure 4-1). After crossing the Virgin River, the route would climb to the south, turn east between Gooseberry Mesa and Little Creek Mountain, and then southeast on Big Plain. The ascending grade on this segment would be predominantly 1.5% with short stretches of 1.6%. The southern alternate main route would tie into the northern route previously described in the Telegraph Flat area and would then follow the north route alignment to East Clark Bench.

The southern route would not pass through the lease areas near Alton and would require the construction of an additional branch line if these mining operations were to be serviced. This branch could utilize the alignment proposed for the main north route between Alton and Telegraph Flat, or a new line could possibly be designed which would extend northward from the southern route through the Johnson Canyon.

Escalante Mainspur Route

A second northern routing has also been proposed which would depart from the previously described northern alignment as it descended into the Sevier River Valley from Fremont Pass (Figure 4-1). The route would run northward through the Sevier River Canyon to Circleville, turn east paralleling Utah State Route 22, and then turn south along Utah 22 to the vicinity of Escalante Canyon. From Escalante Canyon, the line would require several long tunnels and steep grades through the Escalante Mountains, and would then run eastward through Main Canyon to the town of Escalante, Utah. Two variants would involve a southerly route up Alvey Wash directly to the Utah Power & Light Company lease or another route south along the Straight Cliffs to either Right or Left Hand Collet Canyons. The route would then ascend one of the Collet Canyons on steep grades to reach the Kaiparowits Plateau coal reserves.

south and east towards Virgin, Utah (Figure 4-1). After crossing the Virgin River, the route would climb to the south, turn east between Gooseberry Mesa and Little Creek Mountain, and then southeast on Big Plain. The ascending grade on this segment would be predominantly 1.5% with short stretches of 1.6%. The southern alternate main route would tie into the northern route previously described in the Telegraph Flat area and would then follow the north route alignment to East Clark Bench.

The southern route would not pass through the lease areas near Alton and would require the construction of an additional branch line if these mining operations were to be serviced. This branch could utilize the alignment proposed for the main north route between Alton and Telegraph Flat, or a new line could possibly be designed which would extend northward from the southern route through the Johnson Canyon.

4.23

Escalante Mainspur Route

A second northern routing has also been proposed which would depart from the previously described northern alignment as it descended into the Sevier River Valley from Fremont Pass (Figure 4-1). The route would run northward through the Sevier River Canyon to Circleville, turn east paralleling Utah State Route 22, and then turn south along Utah 12 to the vicinity of Escalante Canyon. From Escalante Canyon, the line would require several long tunnels and steep grades through the Escalante Mountains, and would then run eastward through Main Canyon to the town of Escalante, Utah. Two variants would involve a southerly route up Airey Wash directly to the Utah Power & Light Company lease or another route south along the Straight Cliffs to either Right or Left Hand Collet Canyons. The route would then ascend one of the Collet Canyons on steep grades to reach the Kaiparowits Plateau coal reserves.

Marysvale Mainspur Route

Another route has been proposed to connect the northern portion of the Kaiparowits coal field with a little-used local spur located at the town of Marysvale, Utah, owned by the Denver and Rio Grande Western Railway Company (Figure 4-1). The Marysvale mainspur is not the preferred route at present, even though construction of this line is the shortest of all alternatives, because it does not connect with a mainline at a location that minimizes the distances to other markets in the west and because of the limited number of coal producers it would service.

Southern Utah Branchline Routes 4.3

Two major branch or feeder systems have been proposed for bringing coal from the Kaiparowits Coal Field to the terminus of the UPRR mainline alternative routes in the southern Kaiparowits region (Figure 4-1). The preferred route (Big Sage Route) is 67 miles in length and crosses through the center of most of the coal deposits in the Kaiparowits Coal Field. A northern route (Cottonwood Canyon Route) enters the coal field from the west and crosses only a few of the coal deposits.

The branchline railroad routes are conceived as high-volume, low-to-medium speed systems with a short-haul traffic operation. The branchline would link the mainline gathering yard for the northern mainspur route (preferred location at East Clark Bench, 10 miles southwest of nearest coal mine) with the individual mine spurs and loops (Figure 4-1).

Big Sage Branch 4.31

This feeder line would originate at Mile Post 191 of the mainspur route to East Clark Bench. It would climb east then north past Nipple Bench on maximum grades of 2% against empty trains. The route would pass through most of the known leaseholdings on the Kaiparowits Plateau reserves and would terminate in an area known as Big Sage, which is south and west of the Right Hand Collet Canyon. Construction of this branch would include three major bridges, approximately four 12,500 foot sidings, microwave, signaling, and two tunnels totaling 7,700 feet in length.

This line was developed from topographic maps, and should be considered preliminary and subject to change following field investigation and more comprehensive study. More recent reconnaissance indicates that one 4,700 foot tunnel might be eliminated by relocating a portion of the line and by making a large cut.

4.32 Cottonwood Canyon Branch

This line departs from the mainspur on Telegraph Flat (Figure 4-1). The line is identical to the mainspur between Telegraph Flat and East Clark Bench for 17 miles to the entrance of Cottonwood Canyon on a maximum 1.5% grade. Passing through the Cockscomb in a 1,700 foot tunnel into headwaters of Wahweap Valley in an easterly direction, the route then turns northeast, ascending Horn Mountain on a 1.5% grade. Winding into Paradise Canyon, the route tunnels through the ridge between Escalante and Paradise Canyons on a 5,220 foot tunnel, then continues on to the Big Sage area.

4.33 Other Branch Lines

The preliminary alignments for these additional feeder lines accessing specific coal reserves have been developed, but have not been refined (Figure 4-1). One line, approximately 33 miles long, would depart from the northern route at near Panguitch, Utah, and run easterly through Red Canyon towards Bryce Canyon. The line would then run north and east along the Escalante Mountains and join the possible alignment of the Escalante main route alternative. Ruling grades on this route through Red Canyon would be 2% eastbound (empty) and 1% westbound (loaded). Ten degree curves would be encountered in the Red Canyon area. This route could also provide a fourth main route alternative in combination with the Escalante alternative.

A second branch line would begin at approximately Mile Post 12.6 of the Big Sage line. This line, approximately 23 miles in length, would extend in a northerly direction and pass just west of the coal reserves on the Kaiparowits Plateau. It would terminate in Squaw Bench area south of Paradise Canyon. This line could be extended eastward to provide better access to the coal leaseholds.

The third branch line would originate at Mile Post 19.8 on the Big Sage branch line and would extend approximately 10 miles in a southeasterly direction before terminating in the Nipple Bench area.

Railroad System Description 4.4

The railroad system for the preferred route from Milford to East Clark Bench to Big Sage is to consist of the structure typically included in a UPRR system. In accordance with existing minimum UPRR specifications, it is to be constructed over a three-year period, and is to be operated in such a manner that will permit the orderly shipment of 40 MMT of coal per year to the UPRR mainline at Milford.

Railroad Physical Plant 4.41

Due to the anticipated heavy tonnage, the proposed railroad would utilize a basic design similar to the Union Pacific Railroad standard for single mainline trackage:

Adverse Grades	1.0% maximum
Favorable Grades	1.6% maximum
Right-of-Way	200 foot width (24.2 acres per mile) additional r-o-w width may be required in areas of large cuts and fills.
Embankment (fill)	32 foot top width of subgrade, 2H=1V slopes.
Excavation (cut)	52 foot top width of subgrade (both of the above include a 10 foot wide service roadway).
Rail	New 133 pounds per yard, 39 foot sections prefabricated with 24 ties.
Ballast	15 inches ballast plus 6 inches of sub-ballast.
Curvature	Maximum - 7°30' on primary route 9°00' on secondary routes
Grades	Spiraled to allow 50 mph and 40 mph respectively. Maximum adverse - 1.0% on primary route (westbound) 1.2% on secondary routes Maximum favorable - 1.6% on primary route (eastbound) 2.0% on secondary routes Compensated through curves at 0.04% per degree of curve.

4.42 Railroad Construction Scenario

The construction of the preferred route mainspur from near Milford to East Clark Bench and the Big Sage Branchline would be in accordance with concentrated procedures normally followed by UPRR. Construction would proceed through certain stages generally in the order listed below:

1. Detailed Engineering: aerial photography; ground surveys; geologic mapping; definition of alignment, grades, grading quantities, drainage structures, costs, right-of-way requirements, profiles; archaeological survey.
2. Permitting: Bureau of Land Management, State of Utah, fee landowners; right-of-way negotiations; other federal agencies.
3. Contractor Activities: grading; drainage structures; ditching; revegetation of disturbed areas; clearance of road bed, stabilization of debris; stockpiling of topsoil; water procurement (for dual control); surface water control (culvert, rip-rap, check dams); maintenance roadway.
4. Order track and materials.
5. Track laying: turnout switch at mainline; track laid on ballast; track raising to grade height; ties tamped; ballast trimmed; track inspection.

The personnel required for grading contracting work would consist of groups of 50 persons utilizing dozers, scrapers, motor graders, compactors, water trucks, haul trucks, and additional support vehicles. Each of these work teams (5 teams in total) will work on a three-year schedule at two-shifts per day in order to permit the shipment of coal in the fourth year. Five railroad company personnel would be required to work with each team of contractor personnel. A staff of ten railroad personnel is necessary for a central coordination facility. Additional personnel will be required for bridge and tunnel construction.

4.43 Railroad Operation Scenario

The potential coal tonnage expected to be produced from several

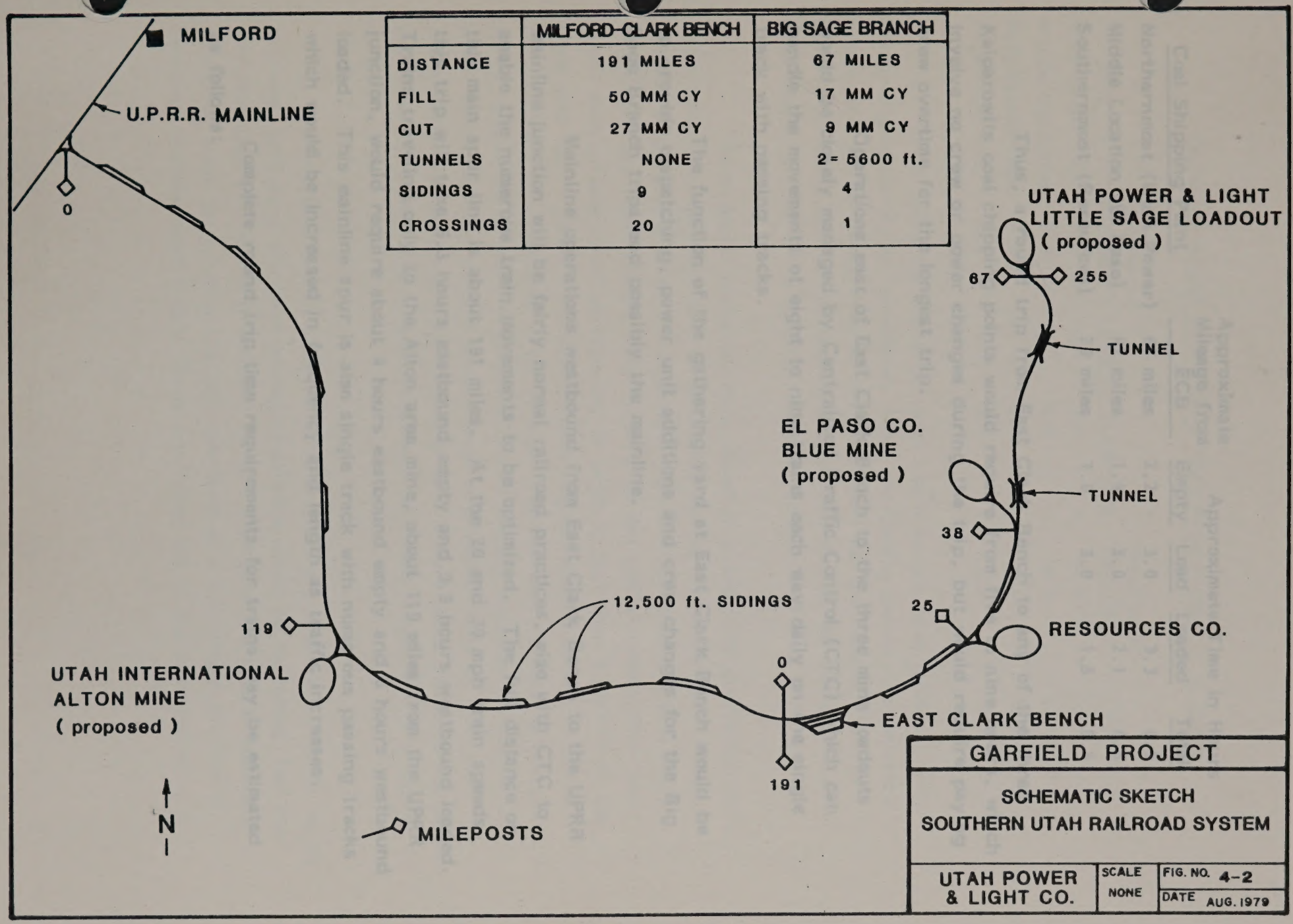
southern Utah coal field projects was described in Section 4.11. A production total of 40 MMTPY from four major mining complexes (UP&L, Resources, El Paso, and Utah International) is suggested as reasonable for this operating scenario.

The previously described preferred route (North - Big Sage) will be utilized, with a gathering yard at East Clark Bench and a staging yard at the UPRR mainline junction near Milford (Figure 4-2). Coal shipments are anticipated from three Kaiparowits coal field mining complexes, and one Alton coal field mine with fairly uniform fast-loading systems at each shipping point.

Unit-train operations are projected exclusively, with trains anticipated to be 100 cars of 100 ton capacity of 10,000 tons each, pulled by four to six locomotives of 3,600 horsepower class. Railroad operations are projected to be of equal capacity throughout the day, week, and year, excepting major holidays (360 days per year). The 40 MMTPY total coal production level would entail 4,000 train round trips annually, or 11.1 train loadings and round trips per day between the four mine complexes. Each loadout would furnish two to six trains per day or one to two trains per shift per mine group. The gathering yard at East Clark Bench would handle more than eight trains per day, once in each direction. The surface mine near Alton is anticipated to ship two to three trains per day directly onto the mainspur line with no additional traffic for the East Clark Bench yard.

Coal loading at the four individual mine coal handling plants would be from large storage systems capable of loading at 4,000 to 5,000 tons per hour onto the 10,000 ton capacity trains. Continuous movement during loading on well-designed loading loops is projected to enable each train to be loaded and turned around in about three hours.

Train speeds on the Big Sage Branchline are projected to average 30 mph when empty and 20 mph when loaded. A round trip for trains going to each shipping location may be projected as follows:



	MILFORD–CLARK BENCH	BIG SAGE BRANCH
DISTANCE	191 MILES	67 MILES
FILL	50 MM CY	17 MM CY
CUT	27 MM CY	9 MM CY
TUNNELS	NONE	2 = 5600 ft.
SIDINGS	9	4
CROSSINGS	20	1

GARFIELD PROJECT

SCHEMATIC SKETCH
SOUTHERN UTAH RAILROAD SYSTEM

UTAH POWER & LIGHT CO.	SCALE NONE	FIG. NO. 4-2
		DATE AUG. 1979

Coal Shipping Point	Approximate Mileage from ECB	Approximate Time in Hours			
		Empty	Load	Loaded	Total
Northernmost (Utah Power)	67 miles	2.2	3.0	3.3	8.5
Middle Location (El Paso)	42 miles	1.4	3.0	2.1	6.5
Southernmost (Resources)	29 miles	1.0	3.0	1.5	5.5

Thus, a round trip from East Clark Bench to any of the three Kaiparowits coal chipping points would require from five to nine hours, which involve no crew or power changes during the trip, but would require paying crew overtime for the longest trip.

Operations east of East Clark Bench to the three mine loadouts would be closely managed by Centralized Traffic Control (CTC) which can handle the movements of eight to nine trains each way daily on the single track with passing tracks.

The function of the gathering yard at East Clark Bench would be to provide dispatching, power unit additions and crew changes for the Big Sage Branch trips and possibly the mainline.

Mainline operations westbound from East Clark Bench to the UPRR mainline junction will be fairly normal railroad practices, also with CTC to enable the numerous train movements to be optimized. The full distance on this main spur line is about 191 miles. At the 20 and 30 mph train speeds, this trip will take 6.3 hours eastbound empty and 9.5 hours westbound loaded. Trains traveling only to the Alton area mine, about 119 miles from the UPRR junction, would require about 4 hours eastbound empty and 6 hours westbound loaded. This mainline spur is also single track with numerous passing tracks which would be increased in frequency and length as traffic increases.

Complete round trip time requirements for trains may be estimated as follows:

Estimated Round Trip Travel Time From Milford - In Hours

Mine	Eastbound Mainline	Branch & Loading	Westbound Mainline	Total
Alton-Utah Int'l	4.0	3.0	6.0	13.0
Nipple-Resources	6.3	5.5	9.5	21.3
Paradise-El Paso	6.3	6.5	9.5	22.3
Sage-Utah P&L	6.3	8.5	9.5	24.3

Crew turns for the Alton mine would be the most difficult, with shifts ending during or immediately after the loading period, necessitating a second crew be transported to the Alton area for the return trip. All other crew turns are within reasonable overtime ranges for fairly efficient crew operations out of two points, Milford Junction and East Clark Bench.

The scenario presented here based on a mid-level production estimate, could be representative of operations during the tenth year of the railroad's operational life. Underground coal mine development is a much longer process than railroad construction, so the coal shipments from any given mine project could be expected to begin at less than 1 MMTPY in the third and fourth years after construction begins, then gradually increase to the 10 MMTPY level over a six to eight year period. Greater production levels, above the 40 MMTPY scenario, would require additions in people, power units, passing tracks, double loops and loadout bins or silos, but the general scheme would not vary significantly. Travel times would be increased somewhat by traffic interference problems, but costs would not vary appreciably.

Many alternatives such as electrification are possible as traffic levels increase, and many alternate operating schemes could be described, but the scenario described above is believed to be a workable and typical situation. Estimated costs for construction and operation of a rail system capable of handling the described traffic levels are presented in the next section.

Railroad Costs 4.5

The costs associated with developing a major railroad system to accommodate the anticipated annual production of coal from the Southern Utah Coal Fields are substantial. These costs are estimated in terms of capital costs to construct the railroad and the anticipated freight rates. The capital cost of the Mainspur from Milford to East Clark Bench is to be borne entirely by UPRR which is reflected in freight rates. The Branchline capital costs are to be borne by the coal producers on a distance proportionate basis which is reflected in lower freight rates for this portion of the railroad.

Construction Costs 4.51

Table 4-1 lists the railroad construction costs estimated by the UPRR for both the main line spur and the branch line to the mines. Section 4.42 gives the expected sequence of construction activities and the rationale behind each step in the railroad design and construction process.

Capital costs of the Big Sage spur construction have been allocated into three portions on the basis of mileage to be shared. The first forty percent of the distance, hence the cost, is divided into three equal parts (shared equally by UP&L, Resources, Co., and El Paso Coal Co.) with one-third allocated to the Utah Power and Light project. The second twenty percent is divided in half (shared equally by UP&L and El Paso Coal Co.) with one-half allotted to Utah Power and Light. The final forty percent is assigned to Utah Power and Light exclusively.

Railroad Costs 4.5

The costs associated with developing a major railroad system to accommodate the anticipated annual production of coal from the Southern Utah Coal Fields are substantial. These costs are estimated in terms of capital costs to construct the railroad and the anticipated freight rates. The capital cost of the Mainspur from Milford to East Clark Bench is to be borne entirely by UPRR which is reflected in freight rates. The Branchline capital costs are to be borne by the coal producers on a distance proportionate basis which is reflected in lower freight rates for this portion of the railroad.

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TABLE 4-1

RAILROAD CONSTRUCTION COST ESTIMATE (UPRR)

(Escalated to Mid-1979 Costs)

Item	Milford-East Clark Bench Mainline		Big Sage Branch	
	Quantities	Cost $ Million	Quantities	Cost $ Million
	191.1 Miles	-	67.0 Miles	-
Engineering	-	$ 12.71	-	$ 3.45
Embankment	50.0 MM cy	143.85	17.3 MM cy	49.74
Excavation	26.9 MM cy	-	8.7 MM cy	-
V-Ditches Above Cuts	254,000 lin. ft.	0.22	137 M lin. ft.	0.12
Tunnels	None	-	2:3,200', 2,400'	18.30
Trackage	1,228,000 ft.	61.47	353,800 ft.	21.60
Sidings	9	7.33	4 @ 12,500 ft.	3.25
Bridges		8.86		10.61
Culverts		12.63		
Right-of-Way	7,000 acres	8.05	1,734 acres	1.99
Fencing	1,228,000 ft.	2.03	353,800 ft.	0.71
Crossings	20	0.42		0.01
Signals		9.57		3.88
Communications		2.06		
Section Facilities		2.21		0.55
Utilities		1.10		11.42
Miscellaneous		15.69		
Scale		0.30		
		$288.50 MM		$125.63 MM

Rail Segment	Distance	Cost	UP&L Share
Big Sage Branchline	67 miles	$125.63 MM	
E.C.B. to Resources	27 miles or 40%	= $ 50.2 MM x 1/3	= $16.7 MM
Resources to El Paso	13 miles or 20%	= $ 25.1 MM x 1/2	= $12.6 MM
El Paso to UP&L	27 miles or 40%	= $ 50.2 MM x 1	= $50.2 MM
			$79.5 MM

Capital costs of the mainline spur (Milford-East Clark Bench) are presented only for information purposes and for support of freight rate estimates. The capital cost of the mainline spur is not directly included in the economic analysis of this project, but is reflected in the rail tariffs estimated by UPRR for delivery from Garfield to various consumer locations. Table 4-2 describes these estimated freight rates.

Freight Rates

Estimation of freight rates for the mainspur and the branchspur was performed by UPRR which presented a rough (± 25%) quotation to UP&L in 1977. These preliminary freight rate figures are the best available data for this specific project and have been escalated to 1979 dollar values (Table 4-2). The escalated estimates are equivalent to recently announced and widely-used ton-mile rates for large volume coal shipments in the western United States and are accepted as realistic for estimating purposes.

These estimates indicate that the capital cost of the Big Sage Branch ($125.6 MM) results in a $1.40 per ton freight rate for coal shipments if all capital costs were borne by UPRR. However, inasmuch as the capital costs are assumed by the coal producers, on a distance proportion basis, freight rates will only reflect the operating costs (which are $1.00 per ton) of the railroad. Where added to the total cost (owning and operating) for the Northern Mainspur ($3.90 per ton), the freight rate for coal shipped from the UP&L Garfield Deposit will be $4.90 per ton or 1.92¢ per ton-mile between the mines and Milford.

TABLE 4-2

ESTIMATED FREIGHT RATES

for LARGE VOLUME MOVEMENTS

from THE GARFIELD PROJECT

Southern Utah Railroad

	UP&L Mines to East Clark Bench	East Clark Bench to Milford	UP&L Mines to Milford	¢/Ton -Mile
Total Cost[1] (fixed & operating) of both Branch and Spur Line	$1.40	$3.90	$5.30	2.08¢
Operating Cost[2] on Branch Line + Total Cost on Main Spur	$1.00	$3.90	$4.90	1.92¢

Complete Coal Delivery Route

Destination	SURR Spur	UPRR Mainline	Total	Mileage	Average ¢/ton-mile
Delta, Utah	$4.90/Ton	$ 2.45/Ton	$ 7.35/Ton	390	1.88¢
Las Vegas, Nevada	4.90	4.70	9.60	520	1.84¢
Victorville, California	4.90	7.80	12.70	700	1.80¢
Long Beach, California	4.90	9.40	14.30	800	1.78¢
Oakland, California[3]	4.90	17.70	22.60	1130	2.00¢

NOTES:

(1) From May 1977 estimate by Union Pacific Railroad Company, escalated 30% to approximate 1979 fuel, labor, and borrowing rates.

(2) Operating costs on 67 mile branch spur estimated at 70% of total costs, or $1.00/ton, leaving $.40/ton or 30% to cover fixed rail bed costs to be covered by shippers.

(3) Oakland freight rate is slightly higher to reflect cost of combined services of Union Pacific and Southern or Western Pacific from SLC to Ogden westward.

TABLE 4-2

ESTIMATED FREIGHT RATES

for LARGE VOLUME MOVEMENTS

from THE GARFIELD PROJECT

Southern Utah Railroad

	UPSL Mines to East Clark Bench	East Clark Bench to Milford	UPSL Mines to Milford	¢/Ton -Mile
Total Cost[1] (Used & operating) of both Branch and Spur Line	$1.40	$3.90	$5.30	3.98¢
Operating Cost[2] on Branch Line + Total Cost on Main Spur	$1.00	$3.90	$4.90	1.92¢

Complete Coal Delivery Rate

Destination	SURR Spur	UPRR Mainline	Total	Mileage	Average ¢/ton-mile
Delta, Utah	$4.10¢/Ton	$ 2.45¢/Ton	$ 7.15¢/Ton	390	1.88¢
Las Vegas, Nevada	3.80	4.70	8.80	520	1.84¢
Victorville, California	4.80	7.80	12.70	700	1.80¢
Long Beach, California	5.40	8.80	14.30	800	1.78¢
Oakland, California[3]	6.30	11.70	22.60	1130	2.00¢

NOTES:

(1) From May 1977 estimate by Union Pacific Railroad Company, escalated 30% to approximate 1979 fuel, labor, and borrowing rates.

(2) Operating costs on 87 mile branch spur estimated at 70% of total costs, or $1.00/ton, leaving $.40/ton or 30% to cover fixed rail bed costs to be covered by shippers.

(3) Oakland freight rate is slightly higher to reflect cost of combined services of Union Pacific and Southern or Western Pacific from SLC to Ogden westward.

5 ECONOMIC ANALYSIS

5.0 ECONOMIC ANALYSIS

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5.5 CONFIRMATION OF ECONOMIC ANALYSIS

5.0 ECONOMIC ANALYSIS

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5.0 ECONOMIC ANALYSIS

LIST OF TABLES

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5. Economic Analysis

Review/Summary of Cost Components 5.1

In Sections 3.0 and 4.0, mine and railroad plans were presented and capital investments estimated for the development of the Garfield project. For the sake of completeness and ease of reference during the development of the coal selling price to achieve the desired return on investment, these data are summarized in the following tables. Tables 5-1 and 5-2 summarize the capital investment required (by year) for the four mine complex, surface facilities, and the railroad spur. Capital expenditures represent the investment in a property and in essence is a trade of present cash outlay for a future inflow of cash.

In this financial analysis, capital investment has been divided into three categories based on the method of capital recovery.

- Depreciable capital includes all mobile equipment, mining equipment, belts, loadout, buildings, engineering, exploration, and other development costs. Depreciable capital is presumed to be recovered in equal per ton amounts over the life of the mine.
- Amortizable capital expenditures are those that have no useful value of their own apart from the project and include earthwork for foundations, railroad grades and highway grades, slopes, and ventilation shafts. Amortizable capital is presumed to be recovered as the tonnage it benefits is produced.
- Working capital is money required to meet early payrolls and to provide cash for supplies and expendables prior to receipt of any income from sale of coal. This has been assumed to be an amount equal to 45 days direct operating expense at full production (between one and two months'

Table 5-1
CAPITAL INVESTMENT SCHEDULE

Cost Category	Initial Investment	Predevelopment Years -4	-3	-2	-1	+1	+2	+3
Predevelopment								
Env. Imp. Stmt.	2.11	1.00	1.11					
Permit Applns.	4.00	1.00	1.00	2.00				
Design Engineering	5.00	1.00	1.00	1.00	2.00			
Drilling	4.00	1.00	1.00	1.00	1.00			
Seam Access								
Mine #1	9.78					3.00	3.00	1.00
Mine #2	17.39						4.00	4.00
Mine #3	16.35							
Mine #4	13.83							
	57.35							
Production Units								
Mine #1	35.61						3.50	10.00
Mine #2	59.38							
Mine #3	59.38							
Mine #4	59.38							
Spares	38.96							1.00
	252.71							
Belts, Haulage, Utilities								
Mine #1	15.54						3.00	7.00
Mine #2	20.95							5.00
Mine #3	20.72							
Mine #4	19.81							
	77.02							
Mine Surface Facilities	24.08							4.38
Access & Utilities	18.16					9.00	9.16	
Coal Handling System								
Mines	7.40							1.85
Overland Belt	23.41					3.34	3.34	3.34
Storage & Loadout	10.80					3.00	5.00	2.80
	41.61	4.00	4.11	4.00	3.00	18.34	31.00	40.37
Mines Total:	486.04							
Railroad (63% of Big Sage Spur)								
Engineering	2.18		0.70	0.78	0.70			25.78
Construction	77.34					25.78	25.78	
Mines & Railroad Totals	565.56	4.00	4.81	4.78	3.70	44.12	56.78	66.15
Working Capital & Spare Parts Inventory	30.00							2.00
Total of Initial & Working Capital	595.56	4.00	4.81	4.78	3.70	44.12	56.78	68.15
Replacement Capital								
Grand Total Annual Capital		4.00	4.81	4.78	3.70	44.12	56.78	68.15

Table 5-1

CAPITAL INVESTMENT SCHEDULE

Table 5-1 (Cont'd.)

Development and Early Production Years									Replacement Capital
4	+5	+6	+7	+8	+9	+10	+11	+12	
				1.00	1.78				57.35 @ 5%/yr. = 2.87
4.00				2.00	1.00	2.39			
4.00	4.00	4.00			1.20	1.60	1.55		
	2.00	2.00	4.00	1.00			2.40	2.43	
10.00	10.00	2.11							213.75 @ 10% = 21.375
7.50	10.00	10.00	10.00	10.00	11.88				
		7.50	12.00	12.00	12.00	12.00	3.88		
			7.50	15.00	15.00	15.00	6.88		
3.00	3.00	3.00	3.00	4.00	4.00	4.00	6.00	7.96	
5.54									77.02 @ 5% = 3.850
10.00	5.95								
	5.00	10.00	5.72						
		5.00	10.00	4.81					
3.14	7.60		4.47	4.49					24.08 @ 5% = 1.204
1.85		1.85	1.85						41.61 @ 5% = 2.080
3.34	3.34	3.34	3.37						
52.37	50.89	48.80	61.91	54.30	46.86	34.99	20.71	10.39	
									Presumed done by Railroad Co.
4.00	6.00	8.00	10.00						None
56.37	56.89	56.89	71.91						
3.14	6.28	9.42	12.56	15.70	18.84	21.98	25.12	28.26	then 31.38 MM annually (6.5% of initial mines capital)
9.51	63.17	66.22	84.47	70.00	65.70	56.97	45.83	38.65	

supplies, labor and overhead) and is added in increments as the mine builds up production. Working capital is presumed to be recovered in the last year of mine life as a lump sum equal to the original amount placed into the operation.

Replacement capital was estimated using nominal life of 10 years for all production units and 20 years for all fixed facilities, leading to 10% and 5% annual replacement expenditures respectively. The weighted average replacement factor derived in this manner is 6.5%. Previous studies performed by Skelly and Loy engineers in which detailed replacement costs were calculated for underground mines yielded annual replacement capital factors ranging from 4% to 7%. Further, experience with active mines and major projects has proven the 6 to 7% annual replacement capital estimate to be valid.

Operating costs are costs incurred during the mining of coal that relate to the application of invested capital in the form of manpower, expendable materials, and government-imposed costs. Also in Sections 3.0 and 4.0, direct operating costs were developed based on manpower required, supplies and materials, power for the mines and the belt system, royalty, insurance and various small taxes unrelated to income. These costs are recapitulated on Table 5-3. It is presumed that regardless of total annual production the per ton cost is the same.

TABLE 5-2
SUMMARY OF CAPITAL INVESTMENT

Year	Initial Capital (million dollars)	Working Capital (million dollars)	Replacement Capital (million dollars)	Total (million dollars)
-4	4.00			4.00
-3	4.81			4.81
-2	4.78			4.78
-1	3.70			3.70
+1	44.12			44.12
+2	56.78			56.78
3	66.15	2.00		68.15
4	52.37	4.00	3.14	59.51
5	50.89	6.00	6.28	63.17
6	48.80	8.00	9.42	66.22
7	61.91	10.00	12.56	84.47
8	54.30		15.70	70.00
9	46.86		18.84	65.70
10	34.99		21.98	56.97
11	20.71		25.12	45.83
12	10.39		28.26	38.65
13			31.38	31.38
14				
15				
16				
17				
18				
19				
20				
21				
22			Total of	
23			22 years	
24			@ 31.38	
25				
26				
27				
28				
29				
30				
31				
32				
33				
34			31.38	31.38
35			0	0
36				
37				
38				
39				
40		-30.00		-30.00
TOTAL	565.56	0	831.66	1397.22

TABLE 5-2

SUMMARY OF CAPITAL INVESTMENT

Year	Initial Capital (million dollars)	Working Capital (million dollars)	Replacement Capital (million dollars)	Total (million dollars)
-3	4.00			4.00
-2	4.81			4.81
-1	4.78			4.78
0	3.70			3.70
1	49.12			49.12
2	56.78			56.78
3	58.15	2.00		60.15
4	52.37	4.00	3.19	59.57
5	50.89	8.00	0.28	63.17
6	48.80	8.00	0.42	56.22
7	41.81	10.00	12.56	64.37
8	54.30		15.70	70.00
9	46.86		18.84	65.70
10	34.99		21.98	56.97
11	20.71		25.12	45.83
12	10.39		28.26	38.65
13			31.39	31.39
14				
15				
16				
17				
18				
19				
20				
21				
22				
23				
24				
25				
26				
27				
28				
29				
30				
31				
32				
33				
34				31.39
35			31.39	0
36				
37				
38				
39				
40		-10.00		-10.00
TOTAL	565.58	0	831.65	1397.22

Total of
22 years
@ 31.39

TABLE 5-3
OPERATING COST SUMMARY

Cost Item	Total Dollar Amount At 18 Million TPY or Method of Calculation	Cost Per Ton	
Direct Salaries	$10,560,000	$ 0.59	
Overhead on Salaries	4,750,000	0.26	
Direct Labor	41,120,000	2.28	
Overhead on Labor	34,500,000	1.92	
UMWA Charge Per Ton	24,930,000	1.385	
Supplies & Materials	88,560,000	4.31	
Power - Underground	7,920,000	0.44	$12.60
Power - Surface	3,060,000	0.17	
Libaility & Disaster Insurance	1,440,000	0.08	
Property (Ad Valorem) Tax	7,850,000	0.44	
Federal Black Lung & Reclamation Fund Taxes	11,700,000	0.65	
Overriding Royalty	1,350,000	0.075	
Federal & State Lease Royalty	8%	-	
Depreciation	41,040,000	2.28	

TABLE 5-3
OPERATING COST SUMMARY

Cost Item	Total Dollar Amount At 18 Million TPY or Method of Calculation	Cost Per Ton	
Direct Salaries	$10,560,000	$ 0.59	
Overhead on Salaries	4,750,000	0.26	
Direct Labor	41,120,000	2.28	
Overhead on Labor	34,500,000	1.92	
UMWA Charge Per Ton	24,930,000	1.385	
Supplies & Materials	80,580,000	4.47	
Power - Underground	7,920,000	0.44	$12.60
Power - Surface	3,060,000	0.17	
Liability & Disaster Insurance	1,440,000	0.08	
Property (Ad Valorem) Tax	7,650,000	0.42	
Federal Black Lung & Reclamation Fund Taxes	11,700,000	0.65	
Overriding Royalty	1,350,000	0.075	
Federal & State Lease Royalty	85	-	
Depreciation	41,040,000	2.28	

Development and Production Schedule

In Table 5-4 is tabulated the expected production rate by year. Four full years of permit preparation, environmental baseline studies, and final design engineering will be required before construction can commence on the mining complex. In the seventh year (third development year), production is expected from the first mine (450,000 tons). Mines are phased in at intervals until full production is achieved in the fifteenth year of the project (eleventh development year).

Development and Production Schedule

TABLE 5-4

DEVELOPMENT PRODUCTION AND SCHEDULE

Tentative Calendar Year	Project Year	Production (tons)	Comments
(1980)	-4		
(1981)	-3		EIS, Permit, Drilling, Design
(1982)	-2		
(1983)	-1		Railroad Construction, Mine Construction
(1984)	+1		
(1985)	2		
(1986)	3	450,000	Mine 1 opens
(1987)	4	1,030,000	First Shipment
(1988)	5	2,040,000	Mine 2 opens
(1989)	6	3,810,000	
(1990)	7	6,030,000	Mine 3 opens
(1991)	8	8,600,000	Mine 4 opens
(1992)	9	11,800,000	
(1993)	10	15,000,000	
(1994)	11	18,000,000	8 years increasing production = 48,760,000 tons.
(1995)	12	18,000,000	
(1996)	13		
(1997)	14		
(1998)	15		
(1999)	16		
(2000)	17		
(2001)	18		
(2002)	19		
(2003)	20		
(2004)	21		
(2005)	22		
(2006)	23		
(2007)	24		
(2008)	25		30 years full production 540,000,000 tons.
(2009)	26		
(2010)	27		
(2011)	28		
(2012)	29		
(2013)	30		
(2014)	31		
(2015)	32		
(2016)	33		
(2017)	34		
(2018)	35		
(2019)	36		
(2020)	37		
(2021)	38		
(2022)	39		
(2023)	40	18,000,000	
	41	12,000,000	4 years declining production = 23,460,000 tons.
	42	6,000,000	
	43	3,000,000	
	44	2,460,000	
		612,216,000	

Financial Criteria

The objective of the financial analysis as defined by UP&L is as follows:

> Selling price of the coal (FOB mine) to assure an 11.6% return on investment (discounted cashflow return on investment method) and the resultant delivered price at plant sites near Delta and Lyndyll, Utah in the Juab/Millard County area.

The 11.6% figure is the weighted average cost of capital considering the three sources of capital available to UP&L (debt, preferred stock, common stock). In the analysis it is presumed that the mining operation would be taxed as a corporation subject to the 46% federal income tax rate.

It is presumed that the sale price determined by the analysis will be accepted by IRS as typical of the Southern Utah coal fields and that since much of the coal will be sold on the open market, that the disposition of the coal will be treated as an arms-length transaction so that the full depletion allowance is granted by IRS. Statutory depletion is assumed to be applicable for tax calculation in all years and is calculated at 10% of gross receipts less royalty not to exceed 50% of net taxable income.

The Utah corporate income tax rate is 6% of net taxable income, which is defined as gross revenue less the typical deductions for operating costs, taxes, depreciation, and depletion. Utah's depletion allowance is defined differently than for federal purposes, being instead 1/3 of gross income less the usual deductions. Thus, the effective state tax rate is 4% of net taxable with no deduction for depletion.

A number of simplifying assumptions were made in the analysis. The project was assumed to be financed at a composite rate of 11.6%. For purposes of calculating the return on investment, all expenditures and all income occur at year's end. It is assumed that during production years,

there is a uniform stream of annual cash flows on a per ton basis. Therefore, such items as annual depreciation and amortization are derived presuming uniform annual per ton write-offs. The investment tax credit is reduced to equal annual per ton credits. Tax law permits rapid write-off methods of depreciation, and investment tax credit is normally taken the year a piece of equipment is placed in service. Use of such techniques produces variable annual cashflows that are beyond the scope of this report.

Financial Analysis

5.4

The Garfield project has been subjected to a preliminary economic analysis using the discounted cashflow return on investment approach, the sophistication of which is believed suitable for this phase of the project. The OCF analysis requires the use of present value factors for the UP&L discount rate of 11.6%. Table 5-5 presents these PV factors for reference.

By definition, the discounted cashflow return on investment is that discount rate that causes the discounted value of a series of disbursements (capital investment) to exactly equal the discounted value of a series of receipts (annual net cash inflow from operations). In this analysis the rate of return on investment and the series of disbursements (capital investment) are known and the unknown is the cash inflow from operations. From the schedule of capital expenditures (Table 5-2), the net present value of the investment for the required discount rate can be determined. Table 5-6 shows these investments present values. The time chosen as the basis for discounting is the beginning of the entire project, the beginning of year -4.

Net Cash Flow Requirement

5.41

To determine annual net cash inflow per ton, each year's cashflow must be discounted using the single payment present worth factor. These thirty-eight present values are summed to achieve the present value of the net cash inflow. Because each year's cashflow is merely the production times the cashflow on a per ton basis, and the cashflow per ton has been assumed constant, this constant factor can be removed from each term in the series. Thus, the dollar amount of the present value of the net cashflow (which is exactly equal to the present value of the capital investment for the 11.6% required return on investment) (Table 5-6) when divided by the sum, over all years, of the product of the present value factor times production in tons (Table 5-7) will yield the net cashflow per ton of production, which is $7.42 per ton:

TABLE 5-5
PRESENT VALUE FACTORS
(11.6% Discount Factors)

Production Year	Years Elapsed	Factor
-4	1	0.8960
-3	2	0.8029
-2	3	0.7195
-1	4	0.6447
+1	5	0.5777
+2	6	0.5176
+3	7	0.4638
+4	8	0.4156
5	9	0.3724
6	10	0.3337
7	11	0.2990
8	12	0.2679
9	13	0.2401
10	14	0.2151
11	15	0.1928
12	16	0.1727
13	17	0.1548
14	18	0.1387
15	19	0.1243
16	20	0.1114
17	21	0.0998
18	22	0.0894
19	23	0.0801
20	24	0.0718
21	25	0.0643
22	26	0.0576
23	27	0.0516
24	28	0.0463
25	29	0.0415
26	30	0.0376
27	31	0.0333
28	32	0.0298
29	33	0.0267
30	34	0.0240
31	35	0.0215
32	36	0.0192
33	37	0.0172
34	38	0.0154
35	39	0.0138
36	40	0.0124
37	41	0.0111
38	42	0.0100
39	43	0.0089
40	44	0.0080

TABLE 5-5
PRESENT VALUE FACTORS
(11.5% Discount Factors)

Production Year	Years Elapsed	Factor
-4	1	0.8960
-3	2	0.8029
-2	3	0.7195
-1	4	0.6447
+1	5	0.5777
+2	6	0.5176
+3	7	0.4638
+4	8	0.4156
5	9	0.3724
6	10	0.3337
7	11	0.2990
8	12	0.2679
9	13	0.2401
10	14	0.2151
11	15	0.1928
12	16	0.1727
13	17	0.1548
14	18	0.1387
15	19	0.1243
16	20	0.1114
17	21	0.0998
18	22	0.0894
19	23	0.0801
20	24	0.0718
21	25	0.0643
22	26	0.0576
23	27	0.0516
24	28	0.0463
25	29	0.0415
26	30	0.0372
27	31	0.0333
28	32	0.0298
29	33	0.0267
30	34	0.0240
31	35	0.0215
32	36	0.0192
33	37	0.0172
34	38	0.0154
35	39	0.0138
36	40	0.0124
37	41	0.0111
38	42	0.0100
39	43	0.0089
40	44	0.0080

TABLE 5-6
PRESENT VALUE OF CAPITAL INVESTMENT

Year	Capital Investment (Million Dollars)	Factor (11.6%)	Present Value (Million Dollars)
-4	4.00	0.8960	3.58
-3	4.81	0.8029	3.86
-2	4.78	0.7195	3.44
-1	3.70	0.6447	2.39
+1	44.12	0.5777	24.49
2	56.78	0.5176	29.39
3	68.15	0.4638	31.61
4	59.51	0.4156	24.73
5	63.17	0.3724	23.52
6	66.22	0.3337	22.10
7	84.47	0.2990	25.26
8	70.00	0.2679	18.75
9	65.70	0.2401	15.77
10	56.97	0.2151	12.25
11	45.83	0.1928	8.84
12	38.65	0.1727	6.67
13	31.38	0.1548	
14		0.1387	
15		0.1243	
16		0.1114	
17		0.0998	
18		0.0894	
19		0.0801	
20		0.0718	
21		0.0643	42.55
22		0.0576	
23		0.0516	
24		0.0463	
25		0.0415	
26		0.0372	
27		0.0333	
28		0.0298	
29		0.0267	
30		0.0240	
31		0.0215	
32		0.0192	
33		0.0172	
34	31.38	0.0154	
35	0	0.0138	0
36	0	0.0124	0
37	0	0.0111	0
38	0	0.0100	0
39	0	0.0089	0
40	-30.00	0.0080	-0.24
		TOTAL	299.96

TABLE 5-6
PRESENT VALUE OF CAPITAL INVESTMENT

Year	Capital Investment (Million Dollars)	Factor (11.6%)	Present Value (Million Dollars)
-4	4.80	0.8960	3.58
-3	4.81	0.8029	3.86
-2	4.78	0.7195	3.44
-1	3.70	0.6447	2.39
-1	44.12	0.5777	14.49
2	56.78	0.5176	29.39
3	68.15	0.4638	31.61
4	59.51	0.4156	24.73
5	63.17	0.3724	23.52
6	66.22	0.3337	22.10
7	84.47	0.2990	25.26
8	70.00	0.2679	18.75
9	65.70	0.2401	15.77
10	56.47	0.2151	12.25
11	46.83	0.1928	8.89
12	36.65	0.1727	6.67
13	37.38	0.1548	
14		0.1387	
15		0.1243	
16		0.1114	
17		0.0998	
18		0.0894	
19		0.0801	
20		0.0718	
21		0.0643	40.55
22		0.0578	
23		0.0516	
24		0.0463	
25		0.0415	
26		0.0372	
27		0.0333	
28		0.0298	
29		0.0267	
30		0.0240	
31		0.0215	
32		0.0191	
33		0.0172	
34	-37.38	0.0154	
35	0	0.0138	0
36	0	0.0124	0
37	0	0.0111	0
38	0	0.0100	0
39	0	0.0089	0
40	-30.00	0.0080	-0.24
	TOTAL		299.96

TABLE 5-7
DETERMINATION OF NET CASHFLOW
TONNAGE DIVIDER

Production x PV Factor		Result
(450,000)	(0.4638)	209,000
1,030,000	(0.4156)	428,000
2,040,000	(0.3724)	760,000
3,810,000	(0.3337)	1,271,000
6,030,000	(0.2990)	1,803,000
8,600,000	(0.2679)	2,304,000
11,800,000	(0.2401)	2,833,000
15,000,000	(0.2151)	3,226,000
18,000,000	1.7854	32,137,000
	TOTAL	44,971,000

TABLE 5-7
DETERMINATION OF NET CASHFLOW
TONNAGE DIVIDER

Production	x PV Factor	Result
(650,000)	(0.4638)	209,000
1,930,000	(0.4155)	428,000
2,890,000	(0.3724)	760,000
3,870,000	(0.3337)	1,271,000
4,030,000	(0.2990)	1,803,000
8,600,000	(0.2679)	2,304,000
11,800,000	(0.2401)	2,833,000
13,000,000	(0.2151)	3,226,000
16,000,000	1.7859	12,127,000
		TOTAL 44,971,000

$$\frac{\text{PV of Total Capital Investment}}{\text{Mine Life (PV Factor for each year)} \times \text{(Production Tonnage for each year)}} = \text{Net Cashflow per ton}$$

$$\frac{\$299.96 \text{ MM (Table 5-6)}}{44.971 \text{ MM tons (Table 5-7)}} = \$6.67/\text{ton}$$

This figure can be verified by multiplying the $6.67/ton Net Cashflow by Annual Production, then discounting each year's cash flow to the present and summing. Table 5-8 presents this method of checking the determination of required net cash flow.

Selling Price Determination

The calculation of an acceptable coal selling price is predicated on the basis that the annual net cash inflow from operations is that amount left from operating income (sale of coal) after all cash costs have been paid. Cash costs are the royalty, operating cost, and taxes. The final step in the determination of the requisite sales price is to work backward through the tax calculations, since state and federal income tax and royalty are unknown and dependent upon sales price. This is a simple job algebraically by letting x represent sales price, expressing unknown costs in terms of x, then deducting all cash costs either explicitly stated in dollars per ton, or implicitly stated in terms of x, and equating that to the annual net cash inflow from operations. Terms are as follows:

(Federal) Royalty = 0.8x

State Tax = 4% x - (.08x + labor + M&S + power + insurance + misc. tax + depreciation + amortization)

Federal Tax = 46% x - (.08 + labor + M&S + power + insurance + misc. tax + state tax + depreciation + amortization) - (x - .08x) (0.10)

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(Federal) Royalty = .0.0x

State Tax = 4% x - (.08x + labor + MSS + power + insurance + misc. tax + depreciation + amortization)

Federal Tax = 46% x - (.08 + labor + MSS + power + insurance + misc. tax + state tax + depreciation + amortization)

- (x - .08x) (0.10)

PV of Total Capital Investment / Mine Life (PV Factor for each year) x (Production Tonnage for each year) = Net Cashflow per ton

$239.95 MM (Table 5-6) / 44.971 MM tons (Table 5-7) = $5.67/ton

This figure can be verified by multiplying the $5.67/ton Net Cashflow by Annual Production, then discounting each year's cash flow to the present and summing. Table 5-8 presents this method of checking the determination of required net cash flow.

TABLE 5-8

PRESENT VALUE OF NET CASH INFLOW FROM OPERATIONS

-4	0	0.8960	0
-3	0	0.8029	0
-2	0	0.7195	0
-1	0	0.6447	0
+1	0	0.5777	0
2	0	0.5176	0
3	3.00	0.4638	1.39
4	6.87	0.4156	2.86
5	13.61	0.3724	5.07
6	25.41	0.3337	8.48
7	40.22	0.2990	12.03
8	57.36	0.2679	15.37
9	78.71	0.2401	18.90
10	100.05	0.2151	21.52
11	120.06	0.1928	
12		0.1727	
13		0.1548	
14		0.1387	
15		0.1243	
16		0.1114	
17		0.0998	
18		0.0894	
19		0.0801	
20		0.0718	
21		0.0643	214.34
22		0.0576	
23		0.0516	
24		0.0463	
25		0.0415	
26		0.0372	
27		0.0333	
28		0.0298	
29		0.0267	
30		0.0240	
31		0.0215	
32		0.0192	
33		0.0172	
34		0.0154	
35		0.0138	
36		0.0124	
37		0.0111	
38		0.0100	
39		0.0089	
40	120.06	0.0080	
		TOTAL	299.96

On Table 5-9 are shown the cost components of the sale price and how they relate to the net cashflow. In Figure 5-1 these costs are shown diagramatically. The selling price F.O.B. mine to achieve an 11.6% discounted cashflow return on investment is $22.87 per ton.

In Section 4.0 of this report, the cost of moving Garfield Project coal by rail to a proposed major power project near Delta and Lyndyll in west central Utah was estimated to be $7.35 per ton. Adding this rail freight cost to the above cost of mining and preparing the coal for utility shipments yields a delivered price of $30.22 per ton at the Lyndyll plant site, or 148¢ per million BTU's (10,200 BTU/lb).

TABLE 5-9

COAL SALE PRICE DETERMINATION

$22.87 per ton	Sale Price
1.83	Royalty
12.60	Operating Cost
2.28	Depreciation
6.16	Net Before Taxes
.25	State Income Tax
5.91	Net Before Depletion
2.10	Federal Tax Depletion Allowance
3.81	Net Taxable for Federal Purposes
1.75	Federal Income Tax
.23	Investment Tax Credit
2.28	Net After Taxes
2.10	Depletion Addback
2.28	Depreciation Addback
$ 6.67 per ton	Net Cash Inflow From Operations

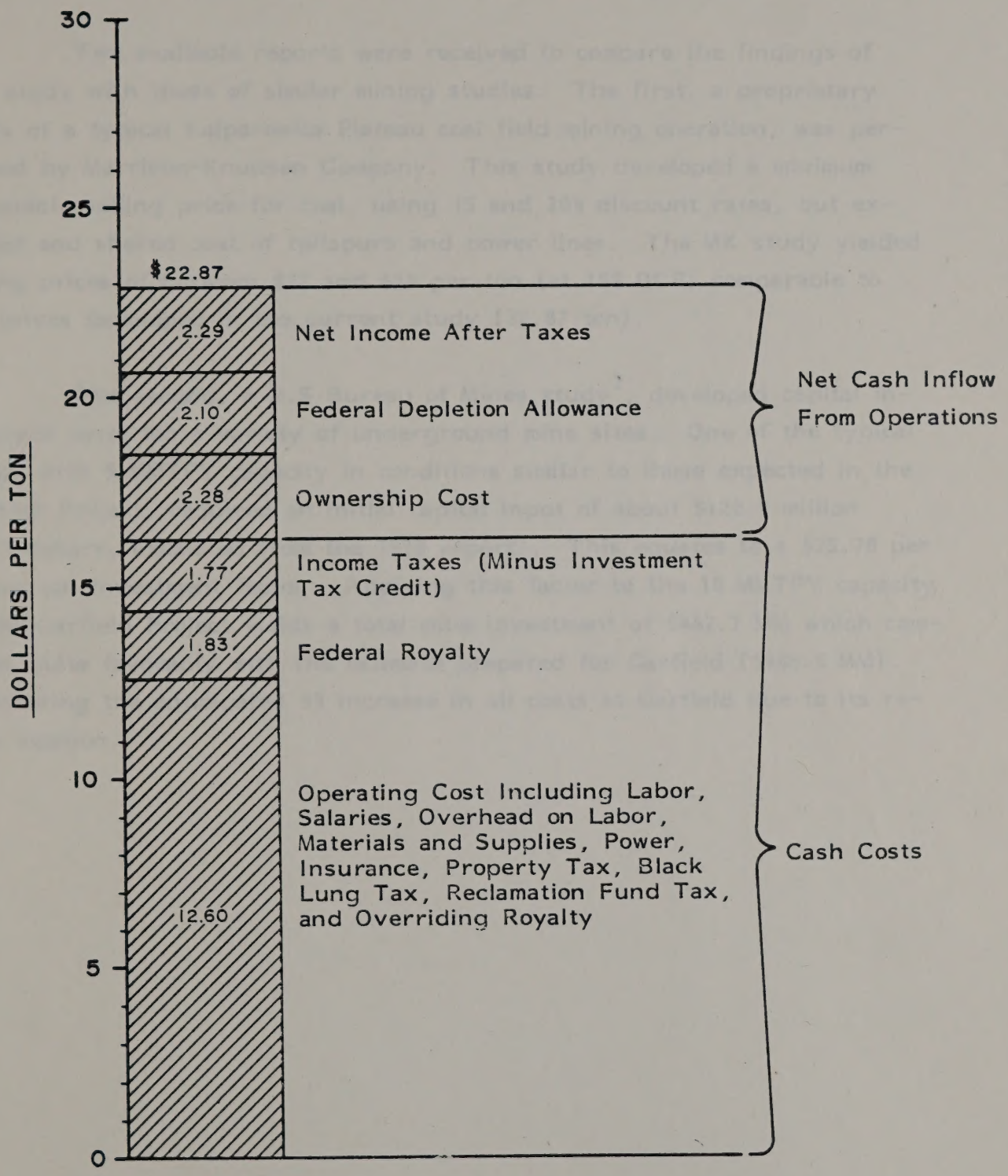
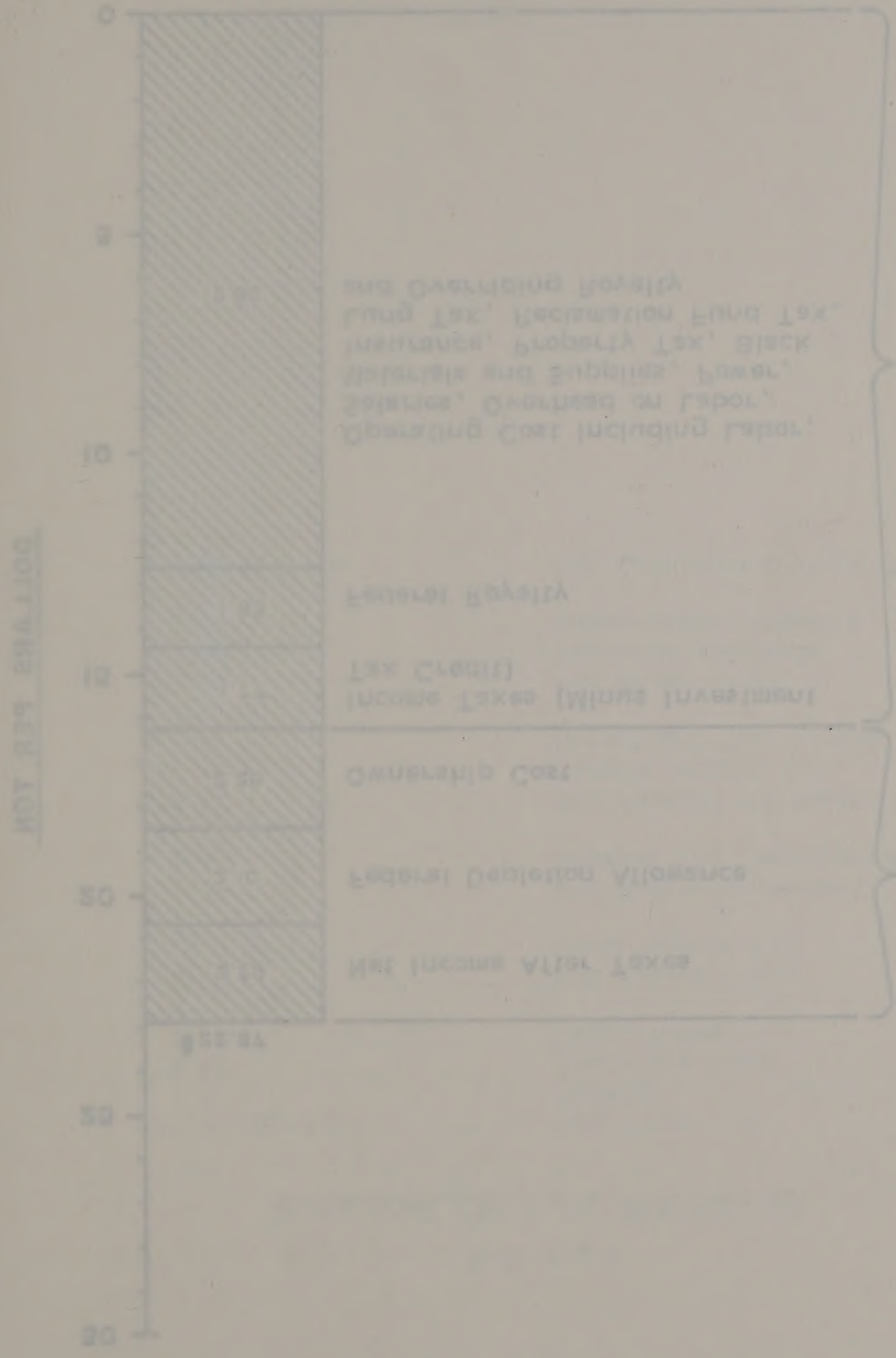


FIGURE 5-1

BAR DIAGRAM ILLUSTRATING COAL SELLING PRICE DETERMINATION


0
10
20
30
40
50
60
Cash Costs
Non-Cash Items
Non-Cash Operations

5.5 Confirmation of Economic Analysis

Two available reports were received to compare the findings of this study with those of similar mining studies. The first, a proprietary study of a typical Kaiparowits Plateau coal field mining operation, was performed by Marrison-Knudsen Company. This study developed a minimum acceptable selling price for coal, using 15 and 20% discount rates, but excluded and shared cost of railspurs and power lines. The MK study yielded selling prices of between $22 and $24 per ton (at 15% DCF) comparable to the prices developed in the current study (22.87/ton).

The second, a U.S Bureau of Mines study[1], developed capital investment costs for a variety of underground mine sizes. One of the typical mines, with 5 MMTPY capacity in conditions similar to those expected in the Garfield Project, required an initial capital input of about $128.5 million (1979 dollars, excalated from the 1975 report). This equates to a $25.70 per annual ton investment factor. Applying this factor to the 18 MMTPY capacity of the Garfield Project yields a total mine investment of $462.7 MM which compares quite favorably with the estimate prepared for Garfield ($486.0 MM), considering the anticipated 5% increase in all costs at Garfield due to its remote location.

[1] IC 8682A: Basic Estimated Capital Investment and Operating Costs for Underground Bituminous Coal Beds.